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J. W. Richards, Ph.D., President.
Curtis E. Whittlesey, Treasurer. E. F. Roeber, Secretary.
Telephone Call: 4700 Bryant.

E. F. Roeber, Ph.D., Editor.
H. C. Parmelee, Western Editor.

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CONTENTS.

Editorial:
Monazite in Idaho ... 655
Condensation of Zinc Vapor ... 655
The Iron Industry's Insatiate Thirst for Capital ... 656
Patent Law and Common Sense ... 657
International Congress of Applied Chemistry of 1912 ... 658
New York Meeting of the American Institute of Chemical Engineers. 658
A Symposium on the Fixation of Atmospheric Nitrogen ... 658
Electric Pig Iron Plant at Frollhättan ... 659
The Western Metallurgical Field ... 659
The Non-Ferrous Metal Market ... 660
The Iron and Steel Market ... 660
Correspondence:
Defects in Refractory Brick. By John C. Bailar ... 661
Some Calculations Pertaining to the Reactions Occurring in a Blast Furnace. By W. D. Brown ... 663
Notes on the Chemistry and Metallurgy of Vanadium. By Warren F. Bleecker ... 666
An Improvement in Hot-blast Copper and Lead Smelting. By Clement H. Mace ... 671
Some Alloys for Permanent Magnets. By C. F. Burgess and J. Aston ... 673
Temperature Drop a True Measure of Thermal Resistance. By Carl Hering ... 676
Zinc Ore Dressing in Colorado. III. By H. C. Parmelee ... 677
Corrosion of Metallic Food Containers. By Edward Gudeman ... 680
The Equilibrium of the System Consisting of Calcium Carbide, Calcium Cyanamide, Carbon, and Nitrogen. By M. de Kay Thompson and Robert H. Lombard ... 682
A Method for Increasing the Calorific Power of Washington Lignites. By H. K. Benson ... 684
The Smelting of Lead Antimony Ores. By A. W. Hahn ... 686
Recent Metallurgical and Chemical Patents ... 687
Synopsis of Metallurgical and Chemical Literature ... 695
Notes on Chemistry and Metallurgy in Great Britain ... 698
Canal Zone Meeting of the American Institute Mining Engineers. 700
Chemists' Building in New York City ... 701
Cement Concrete Vats and Tanks. By Albert Moyer ... 702
A New Radiation Pyrometer ... 703
Lignite as Fuel ... 703
A Modern Air-moistening Apparatus ... 704
A New Caustic Potash and Chlorine Plant at Niagara Falls ... 704
Notes ... 704
Personal ... 705
Digest of Electrochemical U. S. Patents ... 706
New Books ... 706
Book Reviews ... 706

Monazite in Idaho.

It is possible that Idaho will contribute materially to the future domestic production of monazite, valuable for its thoria content and used in the manufacture of incandescent gas mantles. The monazite sands occur there in placer deposits as usual, and their present recovery is an interesting example of retreating material which already has been worked for another purpose. The placers formerly were worked for gold, and in that operation the monazite was concentrated in the fine tailing and went to waste with such gold as escaped. Now these tailings represent the most valuable and easily worked deposits of monazite, although the untouched placer fields will, of course, be worked both for gold and monazite. The practical assurance that monazite existed in commercial quantities in this district was one of the results of the tests on black sand made by the United States Geological Survey at Portland during the exposition in 1905.

The milling of monazite sand involves special treatment in wet dressing, subsequent electromagnetic treatment and cleaning. Screening to remove the coarse gravel is the first step, followed by table concentration and electromagnetic treatment to remove magnetic iron, titanic iron and garnet. The concentrate thus obtained is then cleaned electromagnetically, the product averaging about 95 per cent monazite. The monazite content of the Idaho placer deposits will average at least 0.1 per cent. Determinations of the average thoria content of the Idaho monazite show from 4.75 per cent to 5 per cent, which is a commercial grade. The recovery of gold from the sand should be a great aid in this development if it is to prove commercially successful.

The Condensation of Zinc Vapor.

In the abstracts of Current Metallurgical and Chemical Patents in this issue mention is made of three proposals for the improved metallurgical treatment of zinc ores. The metal zinc, as has often been pointed out in these columns, is among commercial metals "sui generis." In the first place, because of its extreme tendency to form soluble compounds it is a fact that geologically zinc presents divagations and disseminations that no other metal does. Thus there are complex zinc-lead ores, complex zinc-fluorspar ores, complex zinc-copper ores, complex zinc-barytes ores and complex zinc-iron ores, for in geology iron, like the poor, is always with us. Furthermore, in the reduction of the roasted and oxidized zinc concentrate in the spelter plant a marked metallurgical idiosyncracy is noted. This is the fact that the reduction of zinc oxide to the metal by carbon takes place only at a full 100 ° C. above the temperature at which the metal boils in a manner strictly analogous to the manner water boils. Thus zinc is divagatous chemically as well as geologically. This chemical peculiarity is quite natural and related to its geological wandering tendency. For, like those

people whom society cannot quite understand, but of whom society is somewhat in awe, zinc is "unusual"—an adjective used in this connection with a slowly pronounced intonation and with the connotation and suggestion of a vague and vast incomprehensibility.

The attempt of Mr. Frederick W. Gordon to harness by a blast furnace this "unusuality" is a grand and noble idea. Mr. Gordon is an old-time practical engineer and scientist, a man who has designed since 1870 over half a century of iron blast furnaces. He was, if not the innovator, at least a pioneer in the introduction of the water-cooled tuyere and the water-cooled "bosh-plate." Mr. Gordon 20 years ago was working out "heat-balance sheets," which was quite unusual at that time. It may be surmised, however, that his present attempt to attach a condenser to a blast furnace and to make a sort of glorified and magnified "zinc-stuhl" on the pattern of the zinc-stuhl of the medieval Harz "shaftofen" is the last attempt that ever will be made to reduce and condense zinc in a blast furnace. Zinc vapor does not seem to harmonize with heated air. For the zinc molecule above 500° C. has a strong affinity for the atoms of oxygen, even when those atoms are tied up securely to carbon or hydrogen as carbon dioxide or aqueous vapor. Thus it is difficult to see much chance for success for Mr. Gordon to do what Hempel and others have failed to achieve. While Mr. Gordon's argument as to the condensation of zinc in liquid form in the condenser attached to his blast furnace seems strictly correct if the conditions for such liquid condensation have once been established and are properly maintained in continuous operation, yet it would seem that there is grave danger that before these correct conditions can be established the shaft of Mr. Gordon's furnace will be clogged up with zinc oxide and his condenser will be clogged up with blue-powder and a miscellaneous mess. To be concise his proposal seems like the last charge of the Old Guard in the metallurgy of zinc. As such it is admirable and deserves close attention, whatever may be the practical outcome.

Mr. Hiram W. Hixon is a younger man, but he is also noted as a doer of things in the business of making money out of ores. It was Mr. Hixon who over 20 years ago made copper converting practical at Aguas Calientes and Anaconda. Lately he has reduced to a practical basis for the Mond Nickel Company the converting of a low-grade copper-nickel matte to an "80-percent matte." In turning to zinc-metallurgy, Mr. Hixon has with better judgment, we opine, taken to the electric furnace as the most promising metallurgical means to control the unusuality of zinc. Mr. Hixon's ideas as disclosed in his current patents have to do with the eduction of zinc vapor from the furnace to the condenser, and the charging of the electric zinc furnace. The proposals by such an eminent man are worthy of consideration, especially as a veteran of a proved practical turn of mind applies himself to these practical sides of the problem.

The last of the three proposals is that of Mr. Woolsey McA. Johnson. Mr. Johnson is a much younger man and less well known than either Mr. Gordon or Mr. Hixon, but he has a reputation for originality of conception and perseverance in proving the value of his conceptions. Mr. Johnson comes of a family of engineers, for his grandfather, the late William J. McAlpine, among other notable achievements rendered practical the caisson system of building sub-aqueous foundations. His proposal is to increase artificially by eduction the gases emanating from the reduction of low-grade zinc ores through a porous electrically heated carbon filter and thereby render innocuous such weak oxidizers as carbon dioxide, which would hinder the condensation. Mr. Johnson seems to take a philosophical view of the field and to unite the broad commercial side with the scientific and practical side of the question in a quite effective manner.

These three proposals and the attempts of such other workers as FitzGerald, de Laval, Cote Pierron and Queneau (all of which have been noticed before in this journal) give the impression that the atmosphere in the metallurgy of zinc is supersaturated. It may be safely hoped that some of these ideas will condense into commercial form directly. Indeed, the condensation of zinc vapor on a large scale as well as the condensation of ideas seems to be imminent. With labor conditions such as they are now, the 50-lb. zinc retort accordingly is an anachronism in this twentieth century. We predict that in this age of electricity zinc, the unusual metal, will be tamed by that kindly yet powerful force, the electrothermic effect of the electric current.

The Iron Industry's Insatiate Thirst for Capital.

The changing conditions of demand in the iron and steel industry do not remove its thirst for additional capital. In the great period of expansion in productive capacity which may be said to have started with the famine in material in 1899 enormous quantities of capital were invested. From the present outlook it appears that productive capacity has outrun demand and that it will be several years before there is need of any important increases for the purpose of averting a famine in material, but the demand for capital continues. As the investments of the past 10 years have been largely for the purpose of increasing capacity, the money has naturally been spent largely by mill managers and superintendents who were unusually busy with routine duties. They have done their best, but it would be presumptuous to infer that fully as good results, in the introduction of economies, have been attained as would have been the case if the money had been spent for the sole purpose of effecting economies in operation. There is no need for arguing that matter, for if the newer half of the industry represents a great step forward, the older half has the step still to make, since there has been little capacity definitely abandoned.

It is one of the interesting phases of the tendency in the affairs of men to find a balance, to effect a compensation, that while capital is most easily found when prices are high and profits large the necessity for investing capital in improvements is greatest when prices are low and profits small or entirely absent. If the need for expansion in capacity has been removed for a period of years, it follows that the need for improvements so as to reduce costs at many points is presented. The need of capital is not felt alone in the case of entirely new methods, like the use of the electric furnace for steel refining, the duplex process, or the dry-air blast. Since the days of water-power the steam engine has been in use, and yet of late blooming-mill engines have been introduced which are computed as likely to save their cost in three years, when compared with a type replaced and which nevertheless is not

generally regarded as antiquated. The tin-plate industry in the United States is not 20 years old, yet its methods of driving have undergone great changes. The pioneer American tin-plate manufacturers shrank from the nightmare not then uncommon in Wales of an engine direct connected to the hot mills, with a flywheel speeded up by gearing, but halted gearing the engine down to the mill and connecting it direct. Later the rope drive appeared to solve the problem and became almost general, but the latest tin-plate plant erected contains 12 tin mills direct connected to a twin-tandem compound Corliss engine, the rope-drive principle being relegated to the cold mills. With blast-furnace gas engines for blowing and electric power generation and exhaust-steam turbo-generators making their mark, but still introduced only to a limited extent, the sums which ought to be spent in new power-generating plant are evidently enormous in themselves.

Indeed, the needs for capital run the whole gamut of iron and steel operations. Starting with the coke, it is obvious that the retort oven must eventually supplant the beehive oven, but the moderate showing which the retort oven makes in production relative to the whole is due wholly to expansion; there has as yet been no supplanting. In ore, the early operations on the Mesabi range, where mining began just under the roots of the trees, are playing out one by one and must be replaced by operations involving stripping of 100 ft. or more, frequently involving expenditures running into seven figures for a single operation. In the blast furnace there has just come the thin-wall stack, and if half what is hoped of it proves true this curious device will command its introduction. In the fundamental methods of steel production the following figures, in three years of substantially equal totals, are amply suggestive:

Years.	Pig Iron.	Gross tons. Bessemer Steel.	Open-Hearth Steel.
1906	25,307,191	12,275,830	10,980,413
1907	25,781,361	11,667,549	11,549,736
1909	25,795,471	9,330,783	14,493,936

One Bessemer steel plant during the period covered, but another was built and the trend from Bessemer to open-hearth is too marked to promise full employment for Bessemer capacity unless, indeed, all steel-making capacity is required. We have made only the most cursory survey to suggest and by no means to catalog, but coming down at the end to the electric refining furnace we find a line of work in which even the barest outline of the requirements of the next few years cannot be drawn. If the steel industry sometimes enjoys a princely income, it must at all times spend like a prince.

Patent Law and Common Sense.

Discovery and invention are in the language of patent law to all intents and purposes synonymous. To evaluate in terms of use to society any successful invention is the first principle of patent equity. Of late there has come into greater and we believe more deserved prominence the doctrine that invention depends on the practical effort of the inventor to show and demonstrate the success of his process or device. Broadly speaking, the language of Paley, a commonsense English theologian of the eighteenth century, "He alone discovers who proves," exactly fits the case. An analysis of the essentials of an invention in the light of Paley's maxim is of use to the metallurgical or chemical engineer in whose industry the old order ever yieldeth, giving place to the new. The conception of an invention can come from either a flash of intuitive thought or from a long course of hard reasoning, or it can come from an artistic blending of the two, a combination of the purely feminine and the purely masculine character of mental action. Before he can act efficiently and successfully the inventor must see clearly and think correctly. By the building up of a structure of countless mental bricks, each one of which is a strong unit of the aggregate "see-think-and-act," by proceeding from the simple homogeneous unit to the production of the complex, harmonious, heterogeneous whole, the inventor reduces his invention to practice and his original and novel idea has become a useful reality. In short, the value of an invention is measured by the standard of William James' pragmatism—"Utility is the test of truth." The invention of an inventor is comparable to a play of a writer. The invention of a great inventor is comparable to a play of Shakespeare's.

Shakespeare was often accused of plagiarism, especially in the play of Hamlet, the greatest tragic history of a noble soul brought to ruin by a complex of circumstance and environment that the world has ever read. Just in the same way Edison has been accused of inventing "the perfectly obvious." Shakespeare wrote Hamlet, first adopting it from a play of Kyd, then he rewrote it and greatly improved it. Kyd had taken his plot and a good many of his scenes from a French historical novel. This had been taken in turn from the original Scandinavian story. Shakespeare made use of the forms of tragedy brought to perfection by Marlowe from the old miracle plays of the monks. Finally, he made use of many old phrases, rhymes and conceits, of many old tricks of stagecraft. Looked at in a narrow sense there is hardly anything new in it, and on this perverted view the partial truth "there is nothing new under the sun" has indeed a verisimilitude. But any literary critic, in fact, any man of literary taste and ordinary judgment, would dismiss at once the charge of plagiarism from Shakespeare. Shakespeare did not steal Hamlet.

Shakespeare is comparable to the great inventor, for he was a good actor as the great inventor is a fair mechanic; he was a good playwright with a keen sense of the public's wants, as the great inventor has a keen sense of the needs of the industry; he was fairly successful financially just as the higher type of an inventor like Bessemer, Siemens, Bell and Edison have had their share in the financial rewards. In short, they both unite the real with the ideal in a sensible sort of manner. Shakespeare used inferior and old materials and produced a superior article. Bessemer did precisely the same. Just as Shakespeare was original so is the inventor original who proves his invention by intelligently directed hard work. For as old Dean Paley said, "He alone discovers who proves." In the welter and toil of developing an invention the "merely obvious" is usually discovered with difficulty. In fact, what hindsight calls the obvious is usually the antonym of obvious. For, if the obvious was so eternally simple and infernally apparent, why did not someone else discover and prove its existence and value? This commonsense doctrine is seen in every one of the patent decisions of the Supreme Court of the United States. To be a bit disrespectful, law is, after all, only long-winded commonsense.

International Congress of Applied Chemistry of 1912.

In the following we give a list of the presidents and vice-presidents of the different sections of the International Congress of Applied Chemistry to be held in this country in 1912. In each one the president's name is first given. No officers have yet been elected for section 11a on law and legislation affecting chemical industry.

The first meeting of the presidents was held on Nov. 21 at the Waldorf Astoria, under the presidency of Dr. Wm. H. Nichols, president of the congress, and Prof. Morley, honorary president.

As will be seen, there are 23 sections and subsections.

1. Analytical Chemistry: Dr. W. F. Hillebrand, Washington, D. C., and Prof. H. P. Talbot, Boston, Mass.

2. Inorganic Chemistry: Prof. C. L. Parsons, Durham, N. H., and Prof. Charles Baskerville, New York City.

3a. Metallurgy and Mining: Prof. J. W. Richards, South Bethlehem, Pa., and Dr. J. B. F. Herreshoff, New York City.

3b. Explosives: Prof. C. E. Munroe, Washington, D. C.

3c. Silicate Industries: Dr. A. S. Cushman, Washington, D. C., and Dr. Karl Langenbeck, Boston, Mass.

4. Organic Chemistry: Prof. M. T. Bogert, New York City, and Prof. W. A. Noyes, Urbana, Ill.

4a. Colors: H. A. Metz, New York City, and Eugene Merz, New York City.

5a. Industry and Chemistry of Sugar: Dr. W. D. Horne, Yonkers, N. Y., and Dr. F. G. Wiechmann, New York City.

5b. Industry and Chemistry of India Rubber and Other Plastics: Dr. L. H. Baekeland, Yonkers, N. Y., and C. C. Goodrich, New York City.

5c. Fuels and Asphalt: Dr. David T. Day, Washington, D. C., and Dr. F. Schniewind, New York City.

5d. Fats, Fatty Oils and Soaps: David Wesson, New York City, and Dr. Martin H. Ittner, Jersey City, N. J.

6a. Starch, Cellulose and Paper: A. D. Little, Boston, Mass., and Dr. T. R. Wagner, Chicago, Ill.

6b. Fermentation: Dr. Francis Wyatt, New York City, and Dr. Robert Wahl.

7. Agricultural Chemistry: Dr. F. K. Cameron, Washington, D. C., and Prof. H. J. Wheeler, Kingston, R. I.

8a. Hygiene: Prof. W. T. Sedgwick, Boston, Mass., and Dr. L. P. Kinnicutt, Worcester, Mass.

8b. Pharmaceutical Chemistry: Prof. J. P. Remington, Philadelphia, Pa., and Prof. V. Coblentz, New York City.

8c. Bromatology: Dr. W. D. Bigelow, Washington, D. C., and Dr. A. L. Winton, Chicago, Ill.

8d. Physiological Chemistry and Pharmacology: Prof. J. J. Abel, Baltimore, Md., and Prof. W. J. Gies, New York City.

9. Photochemistry: Prof. W. D. Bancroft, Ithaca, N. Y., and R. J. Wallace, St. Louis, Mo.

10a. Electrochemistry: Prof. W. H. Walker, Boston, Mass., and Prof. C. F. Burgess, Madison, Wis.

10b. Physical Chemistry: Dr. W. R. Whitney, Schenectady, N. Y., and Prof. Alexander Smith, Chicago, Ill.

11a. Law and Legislation Affecting Chemical Industry:

11b. Political Economy and Conservation of Natural Resources: Prof. J. A. Holmes, Washington, D. C., and Prof. C. R. Van Hise, Madison, Wis.

New York Meeting of the American Institute of Chemical Engineers.

The third annual meeting of the American Institute of Chemical Engineers will be held in New York City from Dec. 7 to 10. The hotel headquarters are at the Hotel Astor.

The first meeting will be held on Wednesday, Dec. 7, at the Hotel Astor beginning at 9:30 a. m., with an address of welcome by Mr. John Purroy Mitchell, president of the Board of Aldermen of New York City. This will be followed by a business section and by the reading and discussion of the following papers:

Report of the Committee on Chemical Engineering Education, by Dr. F. W. Frerichs.

The Development of the Chemist as an Engineer, by Dr. Fred W. Atkinson.

The Training of Chemical Engineers Which Meets the Requirements of Manufacturers, by Prof. M. C. Whitaker, Columbia University.

The Fitzgibbons Boiler, by Mr. Jerome Alexander.

In the evening a meeting will be held at 8 p. m. at Columbia University, when the following papers will be presented:

Manufacture of Hydrated Lime, by Mr. Richard K. Meade.

Address of retiring president, by Dr. Charles F. McKenna, on The Evolution of Portland Cement Processes.

On Friday, Dec. 9, the morning meeting will be held at the Hotel Astor. After a business session the following papers will be presented:

Manufacture of Lignite Briquettes, by Mr. Henry S. Renaud.

Bleaching Oils with Fullers' Earth, by Mr. David Wesson.

Action of Fruit Juices on Metallic Containers, by Dr. Edward Gudeman.

Vacuum Distilling Apparatus, by Mr. Phillip B. Sadtler.

On the evening of Friday a joint meeting will be held with the New York Section of the American Chemical Society at the Chemists' Club, Prof. C. Baskerville presiding. The following papers will be read and discussed:

The Principles of Sewage Disposal, by Mr. George C. Whipple.

Sewage Disposal in Europe, by Dr. Rudolph Hering.

Sewage Disposal in New York, by Dr. G. A. Soper.

Sanitary Conditions in Their Relation to Water Supplies in the Vicinity of New York, by Mr. N. S. Hill, Jr.

The Unsolved Problems of Sewage Disposal, by Prof. Charles E. A. Winslow.

A quite attractive program of visits and excursions has been arranged.

On Wednesday afternoon, Dec. 7, the Marx & Rawolle Glycerine Refinery, Brooklyn, N. Y., will be visited, where Wood's multiple effect stills are in operation.

On Thursday morning an excursion will be made to the chamber acid plant of the Standard Oil Company at Bayonne, N. J. Through the courtesy of the Standard Oil Company the members of the Institute and their guests will be conveyed by a steam lighter to Constable Hook and return. The boat will leave Pier 4, North River, at 9:30 a. m. Luncheon will be taken at Hugot's Restaurant, New Brighton, S. I. In the afternoon of Thursday a visit will be paid to the candle house of the Pratt Works of the Standard Oil Company and to the grease works of the same company at Blissville, L. I. On the evening of Thursday a dinner will be held at the Hotel Astor.

A Symposium on the Fixation of Atmospheric Nitrogen.

The first meeting of the season of the New York section of the American Electrochemical Society (Mr. E. A. Sperry being the chairman), will be held at the Chemists' Club on Friday, December 16, at 8:15 p. m. The program is a symposium of papers on the fixation of atmospheric nitrogen in all its phases, including the production of nitric acid and nitrates by means of electric discharges through air, the production of calcium cyanamide from nitrogen and calcium carbide, and the synthesis of ammonia. There will be an introductory paper by Dr. E. F. Roeber, a paper by Mr. Charles S. Bradley, the pioneer of the industry in this country, a paper by Mr. John C. Clancy on the use of cyanamide in the metallurgy of gold, and discussions by various other speakers.

Electric Pig Iron.

In connection with our former numerous notes on the large experimental and semi-commercial work now being done in Sweden and Norway on the reduction of iron ore in the electric furnace, we take pleasure in reproducing a photograph of

ELECTRIC PIG IRON PLANT OF THE JERNKONTORET AT TROLLHÄTTAN, SWEDEN.

the outside of the Jernkontoret's electric furnace pig-iron plant at Trollhättan, Sweden, as it appeared under construction this summer. The building to the right is the charcoal storehouse.

According to reports just received the furnace was expected to make pig iron by Nov. 15.

The Western Metallurgical Field.

Cyanidation at the Alaska-Treadwell.

The ores mined on Douglas Island, Alaska, at the properties of the Alaska-Treadwell, Alaska-Mexican and Alaska-United companies, are very low grade, averaging during the year 1909-1910 $2.95 per ton. The ore is treated by stamp-amalgamation-concentration process, the concentrate being shipped by boat to Tacoma for smelting. As the concentrate represents about 42 per cent of the total annual yield, or about $875,000, its metallurgical treatment is a matter of concern. Until 1899 it was treated by chlorination on the company's ground, but subsequent to that time it was contracted to the Tacoma smelter, Tacoma, Wash., where it was sent in 1000-ton lots.

Several years ago the management of the company instituted experiments looking toward cyaniding the concentrate and erected a 10-ton plant, in which the concentrate was ground in a tube mill, run over an amalgamating table, thence to agitating tanks, where it was agitated by compressed air, and finally to filtering apparatus. Precipitation of the gold-bearing solution was effected by zinc dust. Tests on roasting the concentrate indicated that this procedure was unnecessary, as the unroasted material gave excellent extraction.

The success of the experiments is indicated by the fact that a cyanide plant and refinery are among the improvements contemplated at the property. This will eliminate the freight and treatment charges on the concentrate, which have been about $10 per ton, and as the probable cost of cyaniding will not exceed $5 per ton, a saving of $5 to $6 per ton will be gained by the company. This will amount to about $150,000 per annum, which is the proposed cost of the plant. The residue will still have a commercial value when a market is found for it.

Buying and Selling Ore.

It frequently has been intimated that the seller of ore is not well enough versed in the details of the buyer's end of the business to enable him to deal intelligently with the buyer and understand the reasons for treatment charges and schedules of settlement. At the recent meeting of the American Mining Congress Mr. S. E. Bretherton presented some excellent hints to ore shippers, in which he tried to explain the "give-and-take" points in the treatment of smelting ores. He explains the reasons for the manner of taking moisture samples; for penalizing certain elements in the ore and paying a premium for others; for deducting a percentage of lead and copper from the correct assay; for paying a reduced price for the metals; for the variation in treatment price of low and high grade ores and many other points which the shipper should know, and which, if he did know, would create a better feeling between him and the buyer.

Moisture samples usually include more than an average of the fine material, as the coarse contains moisture which would not be eliminated in the time taken for drying. The manner of obtaining an accurate sample of the ore for assay is one that is not always easily explained to the shipper. Hand sampling may be preferred by him, but his particular material may be far better suited to automatic mechanical sampling and vice versa. Again, he does not always understand the disposition of metallics in his ore, sometimes believing that he is defrauded out of their value, when the fact is that the metallics are assayed separately and their value calculated and added to the total value of the ore. In the matter of "salting," the risk is almost altogether on the side of the smelter, as it is far easier to increase the value of a lot by accidentally mixing a small quantity of high-grade ore or metallics, than it is to reduce the value of high-grade ore by the accidental admixture of low-grade material.

Metallurgical losses account for some of the misunderstood points in treatment charges and settlement schedules. High-grade gold ores suffer a greater metallurgical loss than low-grade and the treatment charge is therefore higher. On the other hand, low-grade ores offer less gold on which to make a profit, and the price paid per ounce of gold is lower than with high-grade ores. Silver and lead are sources of large metallurgical loss, not only in smelting, but in the subsequent sale of the product, such as matte, if the smelter has not his own means for refining.

In addition to the losses on the metals, the smelter has to take into consideration the other constituents of the ore, such as silica, barium sulphate, zinc, alumina and arsenic, on which he must ask the seller to share with him the expense attached to putting these materials through the furnace. On the other hand, there are such substances as iron, manganese, lime and magnesia which the smelter is glad to credit to the seller if his ore contains them, as otherwise it might be necessary to purchase such fluxes at a higher cost.

Zinc.

The unquestioned value of the recently discovered deposits of zinc carbonate and silicate at Leadville has directed attention to the inexplicable reason for overlooking them in years gone by. Noted geologists, such as Emmons and Irving, have given the matter their attention, but without offering any satisfactory reason for the oversight, except that at the time the deposits were first opened it was probable that the miner was so intent upon the discovery of gold and silver that he neither regarded the zinc minerals as having value, nor did he have them analyzed to ascertain such a possibility.

Be that as it may, the deposits now known to exist and from which shipments have already been made, are of sufficient importance to warrant the Colorado School of Mines preparing a special bulletin on the genesis and extent of the deposits, the identification of the minerals and the assay of the ore. In view of the specially favorable market for zinc at the present time the mining and reduction of these ores must be of great benefit to the industry.

Colorado.

Mention has been made previously in these columns of the interest being aroused in the rejuvenation of the mining and

metallurgical industries in this State. Recently the Colorado School of Mines issued its *Quarterly* for October, 1910, and devoted the entire issue to an exposition by the faculty of the scope and progress of the mining industry in Colorado.

Special mention is made of the probable success of new processes of ore treatment and their undoubted effect on the production of low-grade and refractory ores that are not being mined at present.

Bearing on the same subject, Mr. Franklin Guiterman, Mr. T. L. Wilkinson, Dr. Victor C. Alderson and Dr. R. D. George recently gave their views on the situation at a luncheon of the Denver Chamber of Commerce. Mr. Guiterman recited the unquestioned decrease in production of smelting ores in the past decade, but found reason to believe that Colorado still offers a profitable field for mining and metallurgical ventures when guided by competent management and professional advice. Mr. Wilkinson reviewed the increase of other methods of ore treatment, including concentration, showing that in 1909 there were produced 165,000 tons of concentrates valued at about $5,000,000, as compared with about 525,000 tons of smelting ores valued at only $6,000,000. He found reason to believe that well-engineered plants for ore dressing would prove successful in this State, but laid stress on the need of employing the best engineering talent obtainable.

Dr. Alderson, president of the School of Mines, spoke briefly of the great value which the school's new ore-testing plant would be to the State in determining the best methods of ore treatment and in avoiding the unwise expenditure of such enormous sums of money as hitherto have been expended on poorly designed mills and ill-adapted processes. Dr. George, as State geologist, spoke strongly on the possibilities of reopening abandoned and partly abandoned camps in the State, where modern methods would accomplish the success which was not possible in former years. Altogether, an increasing feeling of optimism prevails as a result of the investigation of the actual conditions in the State and opportunities are seen which in late years have escaped notice.

Analysis of Zinc Retort Residues.

At the November meeting of the Colorado Section of Western chemists and metallurgists the analysis of zinc retort residues was under discussion. This material offers some special obstacles to the usual methods of analysis on account of the large quantity of carbonaceous matter contained and its general refractory nature. Mr. J. C. Evans outlined the procedure which he had developed in the course of his experience. Silica is determined by making an insoluble residue which is subsequently fused with sodium peroxide, the melt being dissolved in HCl and evaporated in the usual manner. Iron is determined in the combined filtrates from the silica determination, being oxidized by HNO_3, precipitated by NH_4OH, filtered, dissolved in HCl and titrated in the usual manner. If Al_2O_3 is required, it may be determined in one-half of the solution containing the iron before titrating for that element. Lime is determined in the filtrates from the iron and alumina precipitations. Zinc requires special procedure. The sample is disintegrated in a platinum dish with HF and HCl and the former removed by evaporation. The chlorides are taken up with a little HCl, 2 cc of HNO_3 are added and the solution evaporated to dryness. Nitric acid and $KClO_3$ are then added and the solution again evaporated to dryness, from which point the assay is continued as usual. Wet lead and copper are made on the same sample, attacking the material with chlorate and nitric acid and several additions of HCl to decompose the chlorates. Finally 10 cc H_2SO_4 are added and evaporated to fumes. The lead and copper are then determined in the usual manner. For the gold assay the material is first roasted until glowing ceases.

The silver assay is best made by taking such quantity of the ore as will reduce the proper size button, using an excess litharge flux and no flour or nitre.

The Non-Ferrous Metal Market.

The improved condition and advancing market of spelter was the only feature of the metal market for November. There was a slight advance in copper, but there was no feeling of certainty regarding the market and transactions, while fairly large in volume, were confined almost altogether to the needs of consumers. In general the market has been quiet.

Copper.—An advance to 13 cents for Lake copper occurred early in the month, but the differential between Lake and electrolytic remained smaller than usual. The copper statistics were variously construed, but in general may be said to have been favorable as the total stocks for Europe and America were reduced by over 22,000,000 lb. The average quotations were 12⅞ cents to 13 cents for Lake and 12.75 cents to 12.80 cents for electrolytic.

Lead.—The market has been unchanged and steady at 4.40 cents for New York and 4.25 cents to 4.30 cents for St. Louis The volume of business has been satisfactory.

Tin.—Trade in this metal was reduced to a minimum both at home and abroad, being confined to necessities. November tin was available at slightly lower quotations than for October, being about 36¼ cents.

Spelter.—In view of the demand for this metal in the face of a probable shortage in supply, as mentioned last month, the quotations have advanced. Producers are selling sparingly and only small quantities at any one price. New York quotations were 5.95 cents to 6 cents; St. Louis, 5.80 cents to 5.85 cents.

Other Metals.—There was a fair demand for other base metals during November. Aluminium advanced to 23 cents per pound; antimony showed a light demand at prices ranging from 7⅛ cents to 8 cents for various grades; quicksilver experienced a good business at $45 per flask of 75 lb. for large lots, with higher prices for small.

The Iron and Steel Market.

The November elections were unfavorable to the iron and steel market, not so much on account of a prospect being introduced that the tariff will be reduced on iron and steel products, as by reason of the general unsettlement to business which such a political change is generally assumed to involve. The iron industry, long held to be the barometer of general trade, depends even more than it did a few years ago upon the general business activity of the country. There is a closer relation than ever before between mill activity and general activity. Less iron and steel is going into large projects for which financial arrangements are made and against which contracts are placed far in advance of the completion of the work, and much more is going into the ordinary channels of daily consumption, being bought, made and delivered in a hand-to-mouth sort of way, so that iron and steel activity is more closely aligned with the ordinary business activities of the country than ever before. As to the tariff, it is recognized that nothing can be started for a year, and so much else can happen in that year that the iron trade is thinking very little of the tariff.

The iron and steel industry as a whole operated during November at about 70 per cent of its total capacity. Last February it was, roundly speaking, operating at about 95 per cent of capacity, so that it has been losing an average of about 3 per cent a month. The progression has been continuous, but not uniform, there being a very slight contraction in operations in March and April, a rapid contraction in May and June, slower contraction again in July and August, and more rapid contraction again in the past three months.

Prices suffered little change in November. The mills became more convinced than ever that no important gain could be obtained by reducing prices and no important contracts were in negotiation which would tempt cutting. Most finished steel products are subject to cutting at times, or by certain interests.

but, on the whole, there is little more of this than there was one month or two months ago.

There is no prospect of any improvement in general demand in the next two months. Production promises some slight further decrease, as the accumulation of old orders for heavy steel products is further reduced. For several months the current buying has been chiefly in the light products, the output of heavy products being based chiefly upon old orders.

The "unfilled orders" of the United States Steel Corporation showed a decrease during October of 286,153 tons to 2,871,949 tons. This total is the smallest reported, but it is stated that at one time the inter-company orders were included in the total and at some intermediate time this practice was discontinued so that if comparisons were made on the same basis the tonnage last reported would not be the lowest. This, however, is of relatively little consequence, for the reason that on any basis of compilation the present unfilled orders show the lowest relation to existing capacity which has ever occurred. The continued decrease in unfilled orders has been due to the working off of old contracts for heavy products, like rails, line pipe, fabricated steel, etc., and with this loss tapering off and production decreased, the November loss, which will be reported Dec. 10, promises to be small.

Books were opened early in November for rail orders for 1911 delivery, the Erie's order for 33,000 tons being the first placed. This has been followed by some other orders, and inquiries are pending, but, on the whole, the railroads are not ordering large tonnages. The railroads do not contemplate any extensive track laying next year and, some popular notions to the contrary notwithstanding, the annual wear of rails in service does not create a large demand. Available data are somewhat scant, but they indicate that the annual requirements, due merely to the wear of standard sections, aggregate in the neighborhood of 1,000,000 tons.

The Pittsburgh Steel Company will build two 400-ton blast furnaces at Monessen, Pa., where it operates eight 60-ton basic open-hearth furnaces, blooming, billet and rod mills, 200 wire-drawing blocks, 200 wire-nail machines and barb-wire and field-fencing departments. It operates also a hoop and cotton-tie mill at Glassport, Pa., nearby. The company has had outstanding $6,000,000 of stock and $3,750,000 of bonds, made up of $3,250,000 out of an issue of $4,000,000 in 1907 and $500,000 out of an issue of $1,000,000 in 1908, both at 6 per cent. It issues to the holders of the $6,000,000 an additional $1,000,000, for which $1,000,000 cash is paid into the treasury. To Speyer & Company, associated with the Corn Exchange Bank, it issues $7,000,000 of 7 per cent cumulative preferred stock in return for which Speyer & Company are to secure the outstanding bonds for retirement and pay a sum, not stated, into the treasury. For exchange purposes the stock is rated at 99, the first bonds at 110 and the second bonds at 102½, interest being allowed to Dec. 1 and dividends accruing from that date, so that about $4,150,000 preferred stock is involved in the exchange.

Pig Iron.

There has been a little more buying of pig iron, the needs of consumers increasing after an abstention of so many months from active buying. The buying is of a very conservative character and prices have, on the whole, continued to show a weakening tendency, although the actual declines have been light and there have been two cases in which prices have become firmer. The Buffalo furnaces, having become well sold up at $14 and $14.25 at furnace for No. 2 X, have advanced their asking price to $14.50, while basic iron in the valleys is quoted at not under $13.25, furnace, against $13 done in October. The Birmingham market stands at a minimum of $11, Birmingham, this price being made freely for early delivery, but by only a few producers for first quarter, the stand for $11.50 for next year not having been entirely abandoned. In the Philadelphia market $14.75, delivered, has been done on basic iron. The valley market, Pittsburgh delivered prices being 90 cents higher, stands as follows: Bessemer, $15; basic, $13.25 to $13.50; No. 2 foundry, $13.75 to $14; forge, $13 to $13.25; malleable, $13.25 to $13.50.

Steel.

The mills have taken a more decided stand in the matter of basing prices upon Pittsburgh in the case of Bessemer billets and sheet bars. The market had become decidedly unsettled by the naming of prices f.o.b. maker's mill, which led to sharp competition in delivered prices. Bessemer billets are quotable at $23, Pittsburgh, and sheet bars at $24.50, Pittsburgh, with full freight from Pittsburgh to destination. Similar uniformity has not been established in open-hearth steel, owing possibly to the greater number of producers. About the middle of the month a sale of 700 tons of open-hearth sheet bars was made by a Pittsburgh mill at $24 delivered Wheeling, the freight from Pittsburgh being $1. This is regarded as below the present market, which can be quoted at about $23.50, Pittsburgh, for billets and $24, Pittsburgh, for sheet bars. It is interesting to note that the consumer who had the refusal of the open-hearth sheet bars at $24, delivered, offered the order in preference to Bessemer steel producers at the same price, but could not place it. In either sheets or tin plates the preference of the mill is to roll Bessemer steel, open-hearth being used only when the customer requires the material or perhaps when, as in the present instance, a saving can be effected.

Finished Products.

Plates and shapes are being occasionally shaded by about a dollar a ton, and this is true also of wire products. Merchant steel bars are firmer than before. With the opening of order books for next year's delivery the rail mills began to quote per pound, converting their former prices of $28 for Bessemer and $30 for open-hearth into the corresponding figures per pound. It was desired to help discourage the tendency to compare billet and rail prices as, particularly with the more stringent specifications and inspection now in vogue, there is a wide difference in the cost of manufacture. Regular prices are given below, at Pittsburgh, delivered prices being at the Pittsburgh base plus rail freight to destination unless otherwise noted:

Rails, 1.25 cents for Bessemer, 1.34 cents for open-hearth, at mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in., zees and angles 2 in. x 3 in. and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 cents to 1.40 cents, Pittsburgh; 1.30 cents to 1.32½ cents, Cleveland; 1.30 cents to 1.35 cents, Chicago.

Wire nails, $1.70 per keg; plain wire, 1.50 cents; painted barb wire, 1.70 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2.15 cents to 2.20 cents; galvanized, 3.15 cents to 3.20 cents; blue annealed, 10-gage, 1.60 cents to 1.65 cents; painted corrugated roofing, $1.55 to $1.60 per square; galvanized, $2.75 to $2.80.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE

Defects in Refractory Brick.

To the Editor of Metallurgical and Chemical Engineering:

SIR—In METALLURGICAL AND CHEMICAL ENGINEERING for September appeared an article by the writer on the above subject which seems to have been misunderstood. Before explaining what I said I would like to call attention to the difficulty of reproducing any kind of clay work. We read in the history of pottery of one Bernard Palissy, who attempted to reproduce a white enamel which came to his notice. After spending all his fortune and the fortunes of a number of his friends he finally succeeded at the end of some twenty or thirty years.

To those who have had experience in clay work it is not sur-

prising that he should spend so much time and money at it. The surprising thing is that he succeeded at all. An ordinary man certainly would have failed, but Palissy was a beacon light in the history of pottery; a man who did many extraordinary things in his line, but this achievement probably excels all others.

Another great man who devoted his life to pottery was Seger; but Seger, with all his cunning and the financial backing of the German Government, failed to produce a certain pink glaze after many years of patient labor. Again, we read of a noted firm of English potters who had an order for a perfect loving cup. The price was no object, but the cup must be perfect. After spending $1,500 and producing many exquisite cups the manager of the plant decided that the perfect piece of pottery was yet to be made. This was not an attempt to reproduce anything, but simply to make a perfect piece of high-grade ware, which is much simpler.

There recently appeared in some of the magazines and trade journals a note to the effect that the X. Y. Company, after some years of experimenting, had finally succeeded in producing a fair imitation of a certain pottery. This company had spent thousands of dollars and considered itself well repaid when it had produced "a fair imitation" of the pottery in question.

Coming a little closer to refractories, let us consider the brick manufacturers, to whose skill we owe much of the beauty and worth of our cities. How many of them will attempt to match the color of a brick? To be sure, there are certain stock colors that they can and do match with great accuracy, but the experimental kilns are ever producing colors that differ radically from the standard. Sometimes these odd colors are of rare beauty and some of them are reproduced; but a large percentage of our most successful brick makers would no more guarantee to match all of them than a bird fancier would guarantee to match a white blackbird. Can any and all of these rare colors be reproduced? Certainly they can. So can the pink glaze on which Professor Seger spent so much time and money, but many centuries have failed to produce the man to do it.

But there are in the United States to-day manufacturers who can reproduce anything in the line of refractories. Of course, if the sample is made of material that has been carefully hand-picked the consumer must be willing to pay art pottery prices, but if it is made of material that can be produced commercially, then any refractory can be reproduced with certainty and at a price that is not prohibitive.

With this introduction I will try to explain what I said or intended to say in your September issue. I stated that I had once visited a plant where the methods used were not the best, and, on asking the manager about it, was told that the methods in question were the very best for which the consumer would pay; that they were so good, in fact, that all he was getting out of it was the consciousness of giving one hundred cents worth for a dollar and the possibility of educating his customer to the intrinsic value and actual cost of a good refractory.

While speaking of things that have come under my observation let me relate another: One of the plants of the United States Steel Company was having trouble with the refractories used. A manufacturer studied their case and made a refractory to suit their need. The brick in question were 4½ in. x 4½ in. x 10½ in. and were furnished at $40 per 1000. They were made with great care and the manufacturer kept his finger on the pulse of that furnace, if I may use such a figure, ready to change the prescription at any time if it should become necessary. It was not necessary, however. The steel makers expressed themselves as well pleased with the brick, but after a time notified the manufacturer that they could get brick of that size for less money, and as he could not meet the reduced price without sacrificing quality they are now using a cheaper brick made by another firm.

A standard size brick was made here some years ago that sold at the factory for $70 per 1000. They cost in Leadville, where the writer used them, $96 per 1000, and they were worth it. An average firebrick could have been had at that time for $30 and when the Leadville smelters came into the control of non-resident stockholders the cheaper brick were used. Some of these same stockholders are now bewailing the good old days when they could buy good refractories.

Within the last month the writer saw a pile of fire clay that had been weathering for five years. This clay was pronounced by the United States Geological Survey one of the best in the United States just as it was, but the manufacturer, wishing to be ready for anything, is perfecting this lot. The weathering is now about completed, but before the clay is used all impurities will be carefully picked out. Will the consumer appreciate this or will he think the manufacturer is trying to rob him when he asks a moderate return for all this trouble and expense?

Now I do not wish to be understood to say that any manufacturer can and will produce any refractory that is desired. The manufacturers may be divided into three classes. The first class is not competent to produce anything better than an ordinary fire brick. This class will seek other lines when the market demands something better than the present average, which average, however, was made to meet a demand. The second class is composed of business men who prefer to handle the best seller. The product of this class will keep pace with the demand. The third class consists of a few men who follow art for art's sake, who spend their spare time—and money—perfecting a product for which they hope some day to find a buyer. May their hope not be long deferred.

Clay working is not a pure science. There is much empiricism in the practice of the most scientific man that handles clay in any form, whether he be a potter, a maker of refractories or a chemist. The metallurgist with his work on slimes and the Portland cement maker have thrown much light on the subject, and in spite of the fact that for more than forty centuries the ceramist has not been sure of his foundation, he hopes the time is not far distant when he will be able to reproduce, with absolute certainty, anything, whether it be the most delicate glaze, the rarest building brick, the most refractory fire brick or the toughest abrasive. He has not yet reached that degree of perfection; neither has the paint maker, the Portland cement man, the metallurgist nor a host of others.

A short time ago the writer stood on top of an iron furnace. Carloads of ore, flux and fuel were dumped into the furnace in rapid succession. Rivers of molten iron flowed out a hundred feet below. Thousands of tons of raw material were changed into finished products in this plant every day with scarcely a touch of human hands. A few minutes' walk took us to one of the best zinc smelters in the world. Here we saw men laboriously roasting ore by hand, after which it was mixed with carefully prepared fuel and a few shovelfuls put into an expensive clay retort, placed in the furnace and, at the expense of more time and fuel, a little zinc was distilled off. In spite of this outlay of labor and capital the output was only a little over 85 per cent. As we looked across the valley at the majestic stacks of the iron works it seemed as if the wheels of progress had suddenly turned backward many centuries. To the zinc men, however, the outlook was different; they claimed that their output was the greatest that had ever been produced. They did not exalt themselves, however, above their fellows, but admitted that their record-breaking output was due to the superior refractories which they were using.

Returning to refractories, the writer is still optimistic enough to believe that the time is near when the consumer and manufacturer will get together with a full knowledge of the needs of the one and the difficulties of the other, much to the benefit of both.

Golden, Cal. John C. Bailar.

Some Calculations Pertaining to the Reactions Occurring in a Blast Furnace.

BY W. D. BROWN.

By a comparison of the volumes of the constituent gases of blast-furnace gas much is learned that is not readily seen when, as is usually the case, calculations are based on the weights of the constituents of the gas. Moreover, the calculations are much simplified if based on the volumes or percentage volumes of the constituent gases. This follows from Avagadro's law that a given volume of any gas contains the same number of molecules as the same volume of any other gas; and, since reactions of gases are between molecules, it follows that the volumes of gases entering any reaction bear a simple ratio to each other. For example, we have the union of carbon monoxide and oxygen to form carbon dioxide, $2CO + O_2 = 2CO_2$. From this it is seen that two molecules or volumes of carbon monoxide combine with one molecule or volume of oxygen to form two molecules or volumes of carbon dioxide.

Extent of Oxidation of the Carbon in the Furnace.

It is believed that a quantitative investigation as to the amount of carbon existing in the gas as carbon dioxide may throw some light on the reactions occurring in a blast furnace.

It is a well-known fact that any carbon dioxide produced at the tuyeres is quickly reduced to carbon monoxide, part of which, on rising through the charge in the furnace, reduces iron oxide, the final reaction being as follows: $Fe_2O_3 + 3CO = 2Fe + 3CO_2$. From this it is derived that 111.7 tons of iron are reduced by the carbon monoxide containing 36 tons of carbon, or that 1 ton of iron requires for its reduction the carbon monoxide containing 0.322 ton of carbon; the carbon dioxide formed likewise contains 0.322 ton of carbon. Pig iron contains about 93 per cent of iron and the carbon dioxide formed from the reduction of the iron in 1 ton of pig iron contains 0.300 ton of carbon. Iron may also be reduced by solid carbon, the carbon oxidizing to carbon monoxide. This carbon monoxide may or may not reduce other particles of iron, the iron reduced by carbon monoxide being here considered regardless of the source of the carbon monoxide.

The carbon dioxide formed during the reduction of iron by carbon monoxide is, however, not the only source of carbon dioxide in the gas. It is formed when the carbonates of the charge are decomposed, but most of it is reduced by solid carbon to carbon monoxide ($CO_2 + C = 2CO$). There is yet another source of carbon dioxide in the gas; it is formed when carbon is deposited, the reaction being $2CO = C + CO_2$.

It would seem that with these three uncertain sources of carbon dioxide little can be learned. Yet it is readily seen that if the deposited carbon reduces iron the result both as to the resultant gas and as to the amount of heat liberated is the same as if the iron were reduced by carbon monoxide. This is shown by the following reactions:

Reduction by CO.	Carbon Deposition and Reduction by Deposited Carbon.
$2Fe_2O_3 + 6CO = 4Fe + 6CO_2$	$6CO = 3CO_2 + 3C.$ $2Fe_2O_3 + 3C + 3CO_2 = 4Fe + 6CO_2.$

The final products are the same in both cases. In fact, if some of the deposited carbon impregnates the pig, or is burnt before the tuyeres, or reduces carbon dioxide from the flux to carbon monoxide, or decomposes water, and the same quantity of solid carbon of the coke reduces iron, the result is the same as if iron were reduced by carbon monoxide. In reality no matter what the source of carbon dioxide it shows the extent of the oxidation.

By a comparison of the carbon contained in the carbon dioxide formed per ton of pig iron with the amount required if reduction is by carbon monoxide we may learn with what degree the reduction is made by carbon monoxide. The total compounds of carbon found in the top gas are derived from the total carbon gasified and their percentage by volume represents this weight of carbon. This, after deducting the carbon lost in flue dust and in "slips," is the sum of the carbon in coke (c) and in limestone and other carbonates (s), less that entering the pig (p), and may be represented by $(c + s - p)$. Now, since each molecule of each of the compounds of carbon found in blast-furnace gas contains but one atom of carbon and any given volume of any of the carbon compounds contains the same weight of carbon as does the same volume of any other carbon compound found in blast-furnace gas, it follows that the weights of carbon gasified are in proportion to the volumes of the resultant compounds of carbon. Therefore, *the ratio of carbon dioxide to the sum of the carbon compounds is the ratio of carbon in carbon dioxide to the carbon gasified.*

For years the ratio of carbon dioxide to carbon monoxide has been looked upon as an indication of the reduciton going on in a furnace. A more definite indication of the working of the furnace is the ratio of carbon dioxide to the sum of the carbon compounds. This, the percentages by volume of the gases being represented by their formulas, is expressed as $\frac{CO_2}{CO_2 + CO + CH_4}$ and, as shown in the preceding paragraph, is the ratio of carbon existing as dioxide, the higher state of oxidation, to the carbon gasified; and from it we may determine the actual weight of carbon existing as dioxide per ton of pig iron by multiplying by the weight of carbon gasified $(c + s - p)$. It is readily seen that the greater the amount of carbon oxidized to carbon dioxide the more heat is liberated in the furnace, and, what is of more importance, the more carbon has descended in the furnace to unite eventually with air before the tuyeres, for the presence of carbon dioxide means that carbon of the coke has escaped oxidation by carbon dioxide or iron oxide, or carbon has been added to the charge by deposition. By a comparison of the carbon existing as dioxide, as found above, with the carbon required as carbon monoxide to reduce the iron in a ton of pig iron, we may arrive at a concrete knowledge of the reduction. This is best shown by example.

Example I.

Carbon dioxide in gas (CO_2)	12.3 per cent
Sum of carbon dioxide, carbon monoxide and methane ($CO_2 + CO + CH_4$)	38.3 per cent
Ratio of carbon in higher state of oxidation to carbon gasified $\frac{CO_2}{CO_2 + CO + CH_4}$	.3212
Carbon gasified per ton of pig iron	.898 ton
Carbon in higher state of oxidation per ton pig iron $.898 \times .3212$	.288 ton
Carbon dioxide formed from reduction by carbon monoxide of the iron in 1 ton of pig iron	.300 ton

It is seen here that the carbon existing as dioxide is sufficient to have reduced by carbon monoxide but 96 per cent of the iron. This does not show that the 96 per cent of the iron was actually reduced by carbon monoxide, but that more of it could not have been so reduced, and that at least 4 per cent of the iron was reduced by solid carbon of the coke; and if more than 4 per cent of the iron were reduced by solid carbon this amount of carbon escaped oxidation by the carbon dioxide of the limestone. The result as to the gas and the amount of heat liberated in the furnace is the same as if reduction were by carbon monoxide and the carbon dioxide of the limestone were reduced to carbon monoxide by solid carbon of the coke.

Example II.

In an experiment made by Mr. R. J. Wysor in conjunction with the writer (METALLURGICAL AND CHEMICAL ENGINEERING, July, 1910) we have conditions showing more carbon existing as dioxide than is sufficient to have reduced all the iron by carbon monoxide.

Carbon dioxide in gas ..... 13.3 per cent
$(CO_2 + CO + CH_4)$ in gas ..... 38.5 per cent
Ratio of carbon in carbon dioxide to carbon gasified, $\frac{13.3}{38.5}$ ..... .3455
Total carbon gasified per ton of pig iron..... .9725 ton
Carbon existing in gas as CO_2 per ton of pig iron $.9725 \times .3455$ ..... .336 ton
Carbon as CO required to reduce iron in a ton of pig iron ..... .300 ton
Carbon as CO_2 from other sources..... .036 ton

Thus in this test it is seen that the carbon existing as dioxide is more than sufficient to have reduced all the iron by carbon monoxide. Possibly some of this carbon dioxide may have come from the limestone unchanged and the iron reduced by solid carbon of the coke or by deposited carbon, but the result is the same as if the iron were reduced completely by carbon monoxide. There is, moreover, a surplus of carbon as dioxide which may have come from the limestone unchanged or from carbon deposition when the deposited carbon was not utilized in the reduction of iron.

Carbon Burnt before the Tuyeres.

In Gordon's "Translation of Gruner's Studies of Blast-Furnace Phenomena" the ideal working of a blast furnace is set forth as being attained when reduction is made by carbon monoxide alone. Reference is later made to the ideal working of the furnace as being attained when all the carbon of the fuel reaches the tuyeres. It does not follow, however, that all the carbon is burnt before the tuyeres even if reduction is entirely by carbon monoxide, for the carbon may be utilized in the furnace in other ways. Rather is the proportion of carbon burnt before the tuyeres an indication of the general working conditions of the furnace, for, as has just been said, it is influenced by other factors than the manner of reduction. Moreover, the amount of carbon burnt before the tuyeres is all important, for it is the sole source of heat in the smelting zone, and upon the quantity of heat developed here depends the grade of the pig iron produced.

A distinction should be made between the amount of carbon burnt before the tuyeres by dry air and by air containing moisture, as it is only by the former that heat is developed, in the latter case the moisture absorbing heat when decomposed by the carbon. In fact, the ratio of carbon burnt before the tuyeres by the dry air of the blast to the total carbon gasified (including the carbon in limestone) is an indication of the general working of the furnace. This follows from the fact that increasing the limestone, and therefore the amount of carbon used in the reduction of carbon dioxide, decreases the carbon burnt before the tuyeres and increases the total carbon gasified; the ratio decreases with increase of limestone, but not at the same rate as does the ratio of carbon burnt before the tuyeres to the carbon in coke. Moreover, this ratio, as will be shown, is readily calculated from the gas analysis alone.

In a blast furnace the carbon obtains its oxygen from the air and from the charge. The oxide of carbon formed when the carbon is burnt in air, i. e., before the tuyeres, is accompanied by nitrogen; this is the only source of nitrogen (disregarding the negligible quantity in the coke), and none is retained in the furnace. Therefore, by a comparison of the volume of nitrogen and the volume of the compounds of carbon in the top gas, the ratio of carbon burnt before the tuyeres may be found.

Dry air, a mixture of oxygen and nitrogen in the ratio by volume of 1 to 3.78, may be written $(O_2 + 3.78\ N_2)$. The action of dry air on carbon may be written thus: $2C + O_2 + 3.78\ N_2 = 2\ CO + 3.78\ N_2$. From this it is seen that one volume of oxygen, accompanied by 3.78 volumes of nitrogen, acts on carbon with the formation of two volumes of carbon monoxide, accompanied by the 3.78 volumes of nitrogen; the carbon monoxide is then $\frac{2}{3.78}$ or 0.529 of the nitrogen. The air supplied to a blast furnace is, however, never absolutely dry. For purposes of calculation (m) represents the ratio of the volumes of moisture and dry air in the moist air mixture, or the fraction of a cubic foot of moisture accompanying a cubic foot of dry air. Air may then be written $(O_2 + 3.78\ N_2 + mH_2O)$. The action of water on carbon is written thus: $C + H_2O = CO + H_2$, from which it is seen that one volume of water vapor acts on carbon forming one volume each of carbon monoxide and hydrogen. Then (m) volumes of water vapor cause the formation of (m) volumes of carbon monoxide. The action of moist air on carbon at the tuyeres is as follows:

$$(2 + m)\ C + O_2 + 3.78\ N_2 + mH_2O = (2 + m)\ CO + 3.78\ N_2 + mH_2$$

The products of partial combustion with moist air are, therefore, found to be $(2 + m)$ volumes of carbon monoxide and (m) volumes of hydrogen to 3.78 volumes of nitrogen. The carbon monoxide formed from moist air at the tuyeres is therefore $\frac{2 + m}{3.78}$ times the nitrogen.

The percentages of the constituent gases are here represented by their molecular formulas. Then the carbon monoxide formed at the tuyeres is $\frac{2 + m}{3.78}$ N_2 per cent of the top gas. The total compounds of carbon in the gas are $(CO_2 + CO + CH_4)$ per cent of the top gas, and represent a definite weight of carbon; namely, the total carbon gasified in the furnace $(c + s - p)$. Since the weights of carbon gasified are in proportion to the volumes of the resultant compounds of carbon the carbon monoxide burnt before the tuyeres $\frac{2 + m}{3.78} \times N_2$ divided by the compounds of carbon in the gas $(CO_2 + CO + CH_4)$ gives the ratio of carbon burnt before the tuyeres to the total carbon gasified. The *carbon burnt before the tuyeres by moist air is, therefore,* $\frac{2 + m}{3.78} \times N_2 \div (CO_2 + CO + CH_4) \times (c + s - p)$. The carbon monoxide formed at tuyeres by dry air being 0.529 of the nitrogen, *the ratio of carbon burnt before the tuyeres by the dry air to the total carbon gasified is:* $0.529 \times N_2 \div (CO_2 + CO + CH_4)$, or is 0.529 of the nitrogen divided by the sum of the carbon compounds present. The weight of carbon burnt before the tuyeres by the dry air of the blast is then $0.529 \times N_2 \div (CO_2 + CO + CH_4) \times (c + s - p)$.

Example III.

In the experiment mentioned in Example II the carbon burnt before the tuyeres was determined as follows:

Moisture accompanying 1 cu. ft. dry air of blast, m ..... 0.01 cu. ft.
Nitrogen in top gas, N_2..... 57.2 per cent.
Carbon monoxide formed at tuyeres, $\frac{2 + m}{3.78} \times N_2$, $\frac{2.01}{3.78} \times 57.2$ ..... 30.42 per cent.
Total compounds of carbon in top gas $(CO_2 + CO + CH_4)$ ..... 38.5 per cent.
Ratio of carbon burnt before the tuyeres to total carbon gasified, $30.42 \div 38.5$..... 0.790
Total carbon gasified per ton of pig iron..... 0.9725 ton.
Carbon burnt before the tuyeres per ton of pig iron, 0.9725×0.790 ..... 0.7683 ton.
Carbon in fuel per ton of pig iron..... 0.9559 ton.
Ratio of carbon burnt before the tuyeres to carbon entering as fuel, $0.7683 \div 0.9559$.... 0.804
Ratio of carbon burnt before the tuyeres by dry air to carbon gasified, $0.529 \times 57.2 \div 38.5$.. 0.786
Carbon burnt before the tuyeres by dry air of blast, 0.9725×0.786 ..... .7644 ton

It is noticed here that only 80.4 per cent of the carbon in the coke is burnt before the tuyeres, and under Example II it was found that the reduction was by carbon monoxide, supposing the carbon in limestone requires a like quantity to reduce the carbon dioxide driven off from the limestone. In this test the authors found that there was a surplus of hydrogen over and above that driven off from the coke and that derived from the moisture in the blast. This hydrogen must have come from water from some source, and carbon must have been used in liberating the hydrogen. In this test 6 per cent of the carbon in the coke must have been so utilized.

Example IV.

During an experiment on a blast furnace a continuous sample of gas was taken over a period of four and a half hours. Determinations of moisture in the air were also made by psychrometer at intervals of one hour. Coke was sampled continuously; the fixed carbon was determined in the sample, the value obtained being used as the total carbon in the coke.

The data and the calculations follow:

Gas Analyses.		Moisture in the Air.	
		Gr. per Cu. Ft.	Vapor pressure in inches.
CO_2	12.3	7.	0.661
CO	25.8	7.5	0.707
CH_4	0.2	7.	0.661
H_2	4.6	7.	0.661
N_2	57.1	6.	0.555
		Average... 7	0.65

Pressure of dry air and vapor, by barometer.. 29.11 inches.
" " vapor ..... 0.65 "
" " dry air ..... 28.46 "

Ratio by volume of moisture to dry air, $\frac{0.65}{28.46}$ .. 0.023

Nitrogen in top gas.....57.1 % top gas.

Carbon monoxide formed at tuyeres, $\frac{2.023}{3.78} \times 57.1$ .....30.56% " "

Total compounds of carbon in top gas, $(CO_2 + CO + CH_4)$ .....38.3% " "

Ratio of carbon burnt before the tuyeres to total carbon gasified, 30.56 ÷ 38.3.....0.798

Fixed carbon in coke (natural state)....84.1%

Coke per ton of pig iron.....2341 lb.

Carbon in coke per ton of pig iron.....1969 lb.
Carbon in limestone.....11.5%
Limestone per ton of pig iron.....1149 lb.

Carbon in limestone per ton of pig iron..... 132 lb.

Total carbon entering furnace.....2101 lb.
Carbon entering a ton of pig iron..... 89 "

Carbon gasified in furnace per ton of pig iron..2012 lb.
Carbon burnt before the tuyeres per ton of pig iron, 2012 × 0.798 .....1606 "
Ratio of carbon burnt before the tuyeres to carbon in the fuel, 1606 ÷ 1969.....0.816
Ratio of carbon burnt before the tuyeres by dry air of the blast to total carbon, 0.529 × 57.1 ÷ 38.3 .....0.789
Carbon burnt before the tuyeres by dry air of the blast, 2012 × 0.789.....1587 lb.

The Amount of Air Entering a Blast Furnace.

Ordinarily the volume of air blown is obtained from the number of revolutions of the blowing engine and the capacity of the tubs. However, it is well known that there is considerable leakage and it is advisable frequently to obtain the number of cubic feet entering the furnace. This is calculated from the weight of carbon burnt before the tuyeres by the dry air of the blast, as found above.

From the reaction of partial combustion of carbon ($2C + O_2 = 2CO$) it is seen that two atoms of carbon combine with one molecule or two atoms of oxygen. From a knowledge of their atomic weights it is found that 24 lb. of carbon combine with 32 lb. of oxygen, or that 1 lb. of carbon combines with 4/3 lb. oxygen. But it is desirable to state the number of cubic feet of oxygen rather than the number of pounds used. One cubic foot of oxygen at 62° and 30" mercury weighs 0.0843 lb. and 4/3 of a pound will occupy a volume of 15.82 cu. ft. (4/3 ÷ 0.0843), and therefore 1 lb. of carbon combines with 15.82 cu. ft. of oxygen. The nitrogen in the air is 3.78 times the volume of oxygen, or the dry air is 4.78 times the oxygen. Then 75.62 cu. ft. (15.82 × 4.78) of dry air are necessary for the combustion of 1 lb. of carbon, and the number of cubic feet of dry air entering the furnace is found by multiplying the number of pounds of carbon burnt before the tuyeres by 75.62.

The number of cubic feet of moist air may be obtained by multiplying the above by the fraction of a cubic foot of moisture accompanying 1 cu. ft. of dry air and adding the product to the number of cubic feet of dry air.

Example V.

Carbon burnt before the tuyeres by dry air of blast per ton of pig iron.....1712 lb.
Dry air at 62° and 30" mercury necessary for partial combustion of 1 lb. of carbon.... 75.62 cu. ft.
Dry air entering furnace per ton of pig iron, 1712 × 75.62 .....129,461 " "
Moisture per cubic foot dry air..... 0.01
Moisture in blast per ton of pig iron..... 1,294 " "

Moist air at 62° and 30" mercury per ton of pig iron .....130,755 cu. ft.

The Volume of Gas Coming Off from the Furnace.

The volume of dry gas may be found from the volume of dry air in the blast by comparing the percentages of nitrogen in the air and in the gas, as the volumes of dry air of blast and of dry gas vary inversely with the percentages of nitrogen contained therein. If the volume of dry air is calculated from the weight of carbon burnt before the tuyeres, the volume of gas derived from it should equal that obtained by the following method, based on the ratio of the carbon compounds in the gas to the carbon gasified.

One cubic foot of carbon dioxide, at 62° and 30", weighs 0.11591 lb. and 1 cu. ft. of oxygen weighs, under similar conditions, 0.08431 lb. Now, the molecules of carbon dioxide (CO_2) and of oxygen (O_2) contain the same number of atoms of oxygen and the same weight of oxygen. The oxygen contained in a cubic foot of carbon dioxide is then 0.08431 lb. The difference between 0.11591 lb. and 0.08431 lb. or 0.0316 lb. is the weight of carbon in a cubic foot of carbon dioxide. Analogous reasoning applied to the other compounds of carbon in the gas gives the same result—0.0316 lb. as the weight of carbon in 1 cu. ft., at 62° and 30", of the carbon compounds present in the gas. Then, by multiplying the percentage of carbon compounds in the gas by 0.0316 the weight of carbon contained in a cubic foot of the dry gas is obtained. The total carbon gasified being $(c + s - p)$ pounds per ton of pig iron, the number of cubic feet of dry gas is $(c + s - p) \div 0.0316 (CO_2 + CO + CH_4)$.

The gas, however, contains moisture. This is usually stated as (m') grains per cubic foot of dry gas at 62° and 30". By multiplying the number of grains by 0.0030 the fraction of a cubic foot of moisture accompanying 1 cu. ft. of dry gas is obtained, and from it the cubic feet of moist gas per ton of pig iron is readily found.

Example VI.

Total compounds of carbon in the gas.....38.5 per cent.
Weight of carbon in 1 cu. ft. of carbon compounds (at 62° and 30") .....0.0316 lb.
Weight of carbon in 1 cu. ft. of gas, 38.5×0.0316...0.012166 lb.
Total carbon gasified per ton of pig iron.....2178 lb.
Volume of dry gas at 62° and 30", $2178 \div 0.012166$.....
.....179,056 cu. ft.
Volume of dry air entering furnace (Example V)....
.....129,461 cu. ft.
Volume of dry gas, calculated from nitrogen, $129,461 \times \frac{79.1}{57.2}$ .....179,028 cu. ft.

Summary.

The foregoing calculations are presented somewhat in detail and may be tedious to some. The results are here presented as formulas, by which calculations can be readily made.

Carbon gasified in the furnace after deducting the solid carbon driven off from the top of the furnace is the sum of the carbon in the coke (c) and in limestone (s) less that entering the pig, and may be expressed as $(c+s-p)$ pounds or tons per ton of pig iron. The percentages by volume of the constituent gases of the top gas are represented by their formulas; thus, N_2, CO_2, CO and CH_4 represent their percentages in the top gas. The moisture in the blast is represented by (m), the ratio by volume of moisture and dry air, or the fraction of a cubic foot of moisture accompanying a cubic foot of dry air.

Extent of Oxidation of Carbon.

Ratio of carbon as dioxide to total carbon gasified,

$$\frac{CO_2}{CO_2+CO+CH_4}$$

Carbon gasified in furnace per ton pig iron, $(c+s-p)$ ton.

Carbon existing as dioxide per ton of pig iron,

$$\frac{CO_2}{CO_2+CO+CH_4} \times (c+s-p) \text{ ton.}$$

Carbon required as dioxide if all iron is reduced by carbon monoxide, 0.300 ton.

See Examples I and II.

Carbon Burnt Before the Tuyeres.

Ratio of carbon burnt before the tuyeres by dry air of blast to carbon gasified,

$$0.529\,N_2 \div (CO_2+CO+CH_4).$$

Carbon burnt before the tuyeres by dry air of the blast,

$$0.529\,N_2 \div (CO_2+CO+CH_4) \times (c+s-p).$$

Ratio of carbon burnt before the tuyeres by moist air of the blast to carbon gasified, $\frac{2+m}{3.78} N_2 \div (CO_2+CO+CH_4)$.

Carbon burnt before the tuyeres by moist air,

$$\frac{2+m}{3.78} N_2 \div (CO_2+CO+CH_4) \times (c+s-p).$$

See Examples III and IV.

Amount of Air Entering the Furnace, as Calculated from the Carbon Burnt before the Tuyeres.

Dry air, at 62° and 30", necessary for partial combustion of 1 lb. carbon, 75.62 cu. ft.

Dry air entering furnace per ton of pig iron is 75.62 times the carbon burnt before the tuyeres by dry air and is, when $(c+s-p)$ represents pounds, $0.529\,N_2 \div (CO_2+CO+CH_4) \times (c+s-p) \times 75.62$.

Moisture per cubic foot dry air is (m).

Moisture per ton of pig iron is (m) times the dry air entering furnace.

Moist air entering furnace per ton of pig iron is the sum of dry air and moisture.

See Example V.

Volume of Gas at Top of Furnace, Reduced to 62° and 30" Mercury.

Weight of carbon in 1 cu. ft. (at 62° and 30") of CO_2, CO or CH_4, 0.0316 lb.

Weight of carbon in 1 cu. ft. gas, $0.0316\,(CO_2+CO+CH_4)$.

Carbon gasified per ton of pig iron, $(c+s-p)$ lb.

Volume of dry gas, $(c+s-p) \div 0.0316\,(CO_2+CO+CH_4)$.

Moisture in gas reduced to grains per cubic foot dry gas at 62° and 30", m'.

Volume of moisture accompanying 1 cu. ft. dry gas, $0.003012\,m'$ cu. ft.

Number of cubic feet of moisture per ton of pig iron can be found by multiplying the number of cubic feet of dry gas by $0.003012\,m'$.

Number of cubic feet of moist gas is their sum.

See Example VI.

Laboratory, Duquesne Steel Works,
Carnegie Steel Company.

Notes on the Chemistry and Metallurgy of Vanadium.

By Warren F. Bleecker.

Introduction.

The constantly increasing importance of vanadium products in the steel industry, as well as in other branches of manufacture and arts, has stimulated investigation of the future supply of the element, and with the discovery of sufficient quantities of the ores the problem has been turned over to the metallurgist to make available the valuable content desired in the many combinations and forms by the various consumers. The problem is not an easy one, because the chemistry of vanadium is but little understood. Of the metallurgy, written in scientific language, there is no literature to be found, and, with a theoretical working knowledge of the chemistry of the element, the metallurgist is still at a loss to know how to proceed, because almost all writers have disagreed as to elementary facts, even as to what analytical group of elements vanadium belongs. A careful study of all the laboratory data obtainable reveals the fact that the problem is not to be solved in the mere attempt to produce reactions which are analagous to those of other elements, nor by a simple study of general relationship to other elements in the same or related groups. On the other hand, specially designed metallurgical methods may be developed from a more or less incomplete knowledge of the principal vanadium compounds, which methods can be and have been designed and put into operation with success.

The investigations recorded in this paper have extended over a period of two and a half years of continuous work, and while many facts have been definitely established, there is much yet to be done before some of the methods herein described can be used on a large scale without the supervision of an expert.

I have endeavored to divide the paper into three parts, dealing respectively with the chemistry of vanadium, vanadium analysis and some new metallurgical methods. Under the first heading I shall attempt to give such chemical data as will enable the reader who has not made a study of vanadium to understand without difficulty the methods of analysis and extraction that follow. Wherever possible I have examined the original references, and where I have not personal knowledge of the accuracy of experiments or statements I hope to have put the matter in such a manner as not to leave the reader in doubt as to the authority. Only the chemistry of such compounds is given as will have a bearing upon the following subject matter, or have for some reason properties of more than passing importance.

Under the second heading I have tried to discuss all of the useful methods of vanadium analysis, and a great many which are not useful, but important, either historically or because of the weight of authority that is still bearing upon them. I shall make criticisms freely, but not in a spirit of having said the last word. Rather, I am simply trying to state certain difficulties which I have encountered, and in each case I shall give reasons and authority for statements, and in addition try to

develop the correct formulas, factors, etc., by arguments based on facts given under the first heading.

Under the third heading I shall discuss in detail several methods for the extraction of vanadium from various crude ores, and methods of refining and producing compounds upon a commercial scale. I shall give some methods said to be in use with all the detail available. I shall criticise these methods, both favorably and unfavorably, and finally I shall endeavor to discuss my methods fairly, giving points of weakness as well as points of excellence.

In short, the object of this paper is to give information which is intended to be accurate and sufficiently in detail to give the reader an intelligent idea of what is being said so that he may properly judge its value. I hope that mere opinions given here are not taken as statements of fact, but I feel that opinions are valuable, because they may contain at least the germ of an idea, which unexpressed, might be forever lost.

The Chemistry of Vanadium.

Vanadium is a metallic element, never found free in Nature, but always combined with other elements forming complex minerals. It has an atomic weight of 51.2 and a valence of three and five. The specific gravity is given as 5.5, and the metal is not magnetic, nor are any of its compounds, excepting the alloys with magnetic metals.

Metallic vanadium may be prepared by heating the dichloride in a stream of dry hydrogen gas, the dichloride being made by passing dry chlorine gas over crude vanadium obtained by the reduction of vanadic acid with aluminium according to the Goldschmidt process. The vanadium obtained in this manner is said to be practically pure, and is in the form of a steel-gray powder, which is similar in appearance to tungsten metal.

Vogel and Tammann[1] claim to have made metallic vanadium assaying 99.07 per cent V by reducing the pentoxide with aluminium in a crucible lined with magnesite. The assay was made by the method of Rose[2] and the chief impurity was silicon. The remarkable statement is made that no aluminium could be observed in the final product. In view of the work of other experimenters and of my own experiments, I am inclined to believe that some error exists, especially since the method of Rose is not the easiest method with which to obtain accurate assay.

Prandtl and Bleyer[3] experimented with a reducing agent composed of a mixture of metallic calcium and aluminium, and obtained by the reduction of the pentoxide of vanadium metallic reguli which assayed as high as 94.45 per cent metallic vanadium. The purest regulus was very hard, crystalline, not brittle, and took a high polish which was stable. The color was steel-gray, and the metal was insoluble in concentrated hydrochloric acid.

I have reduced vanadium pentoxide by means of the electric arc in the form of a regulus, but the product contained much carbon and considerable oxygen, although not enough to account for all of the vanadium as either the monoxide or carbide.

Metallic vanadium is slowly soluble in sulphuric and nitric acids and in water solutions of the fixed alkalis. It is stable in the air at ordinary temperatures, but rapidy oxidizes upon heating in the presence of air.

Contrary to published statements I have never found vanadium to exhibit electro-positive properties in any solution, but always find it as an anode product, the VO_3 ion even finding its way through a porous membrane from the cathode compartment of a cell, the electrolyte of which is acid.

Vanadium, like antimony, arsenic and uranium, plays a double rôle. It unites with bases to form three series of salts known as vanadates, analagous to the salts of phosphorus—ortho, meta and pyro respectively. It also, when combined with one atom of oxygen, unites with acid radicals to form two series of salts known as vanadyl salts.

Vanadium unites with oxygen to form five oxides: V_2O_5, V_2O_4, V_2O_3, V_2O_2, V_2O, called the pentoxide, tetroxide, sesquioxide, dioxide and monoxide respectively. Of these the latter three are of academic importance only, while the tetroxide of vanadium is met with only in combination with other elements, excepting where it is especially prepared. The pentoxide of vanadium is a very important compound, and it is popularly known and spoken of as vanadic acid. It is known in three distinct physical forms, which are described as yellow, red and crystalline.

Vanadic Acid.

The yellow variety is the one sold by chemical supply houses put up in small phials and labeled "Highest Purity." In this particular instance this means less than 98 per cent. Since the product contains titanium it is difficult to obtain the exact percentage of purity, because, as I shall show later, titanium present in vanadium products is in part reported as vanadium. It is easiest made by the hydro-electrolysis of a fused copper vanadate ingot, and obtained in this way may be 100 per cent pure. It is soluble in water, a saturated solution containing 300 mg to 400 mg per liter, depending upon the position of ammonium metavanadate by heat, and sometimes results from the decomposition of vanadates by mineral acids. This yellow vanadic acid sometimes results from the decompresence of slight traces of mineral acids. Pure solutions usually do not contain quite as much as 300 mg. The water solution is strongly colored yellow, the minutest trace of the dissolved acid being apparent. Although the yellow vanadic acid may be dissolved in mineral acids, the best solvent is either sodium hydroxide, to which a little hydrogen peroxide has been added, or sodium carbonate. Alkalis dissolve the vanadium to a colorless solution, while concentrated sulphuric acid dissolves it to a deeply colored solution, blood red when very concentrated, but blue upon dilution.

The red variety of vanadic acid is the one most commonly met with in the laboratory, and, like the yellow acid, is amorphous and very light. It results from the action of mineral acids upon metallic and alkali vanadates as a rich looking, curdy precipitate which is easily filtered and washed. The vanadate must either be in the form of a dry powder, or in a very concentrated solution in the case of alkali vanadates. The mineral acid may be added in any desired degree of concentration, but too great a quantity will dissolve the vanadic acid completely in most cases. The precipitation is not in any case quantitative, but ordinarily, by careful manipulation, 25 per cent to 30 per cent of the vanadic acid may be removed from copper vanadate by decomposition with sulphuric acid. This reaction will be discussed more fully under a description of the commercial production of vanadic acid. The red vanadic acid deports itself similarly to the yellow variety, except that it is slightly more soluble, and a water solution of the pure salt may contain over 500 mg per liter of the acid. It is soluble in acids, out of which it may be precipitated by boiling if the concentration is right. The product formed in this way is never pure, but always contains a portion of the impurities in the solution.

Dissolved in hydrochloric acid the red oxide sometimes changes to the yellow oxide upon evaporation of the solution, but more often the crystalline form results from this treatment.

The dried oxide feels greasy to the touch, and discolors the skin to a slight extent. It rapidly takes on water to the extent of nearly 10 per cent. This water may be removed by careful heating to about 300 deg. C., a higher temperature resulting in a blackening and finally in fusion and transformation into the crystalline variety. The finely divided powder has an odor similar to that of rotten apples, and is unmistabable. All acid vanadium solutions have this odor as well.

[1] *Zeit. f. anorg. Chem.*, 1909, p. 225. R. Vogel u. G. Tammann. Zur Darstellung des Vanadins auf Aluminothermischen Wege.
[2] Analytical Chemistry, Treadwell-Hall, Vol. II, p. 231.
[3] *Zeit. f. anorg. Chem.*, 1909, p. 217. W. Prandtl u. B. Bleyer. Uber die Darstellung von Vanadin und anderen Metallen nach dem Thermitverfahren.

The crystalline variety of vanadium pentoxide is obtained from either of the other two by fusion in either porcelain or platinum. The mass becomes liquid at about 600 deg. C., and may be easily poured into molds. Upon cooling it decreases in volume about one-fifth, and is quite hard and very brittle. When broken, the ingot shows a remarkable crystalline structure which is in the form of needles arranged parallel and extending inward perpendicular to each surface. The ends of the crystals meet at 45 deg. and a vertical section has the appearance of three pyramids, each with the base extending outward. In certain cases where the vanadic acid is the result of an acid evaporation, the mass cannot be obtained in an ingot, on account of its boiling out of the mold at the time of cooling. No amount of remelting will enable the chemist to get a good pour, the mass acting in the same way each time. The action is probably due to the absorption and expulsion of gas, either oxygen or nitrogen. A familiar example of this phenomenon is that of the absorption of oxygen by pure molten silver and its subsequent expulsion upon cooling.

Small amounts of impurities affect the properties of fused vanadic acid to a marked degree. Potassium salts make the ingot yellowish in color and cause it to absorb water and disintegrate. Titanium makes the ingot extremely hard, while iron makes it brittle. The presence of copper does not seem to have any effect on its physical properties when cold, but it seems to make it more tractable while being melted and poured.

The oxide while molten is a very powerful reagent, attacking all containers, even platinum and fused silica. It attacks silica crucibles with the formation of silicates; it unites with iron to form a crystalline variety of vanadate of iron, which has the appearance of the natural phosphate of iron; it attacks carbon and graphite crucibles with the formation of vanadium carbide. Fused silica, and porcelain, which are attacked but slightly at first, cannot withstand the strains due to the cooling of the thin crust left after pouring. I have found platinum crucibles to undergo no loss after several melts, and again, to lose 75 mg during a single melt. It is possible that this loss may have been due in every case to the presence of some reducing agent, for it is a well-known fact that metallic vanadium will attack platinum very readily.

The thin coating of the oxide left in the crucibles after pouring presents a beautiful crystalline appearance, the crystals being arranged in clusters with radiating elements. This coating, although thin, adheres strongly to the crucible, and is removed only by continued treatment with either strong acids in large quantities or by boiling in a vessel filled with alkali.

Sir William Crookes[4] mentions that platinum crucibles after having been used for the ignition of vanadium pentoxide several times lose their shape, the crucible becoming distorted at the bottom. This is due to the strains set up in the structures of the oxide while cooling, and accounts for the fact that the best porcelain crucibles are shattered during the process of melting in a flame which does not heat the crucible evenly.

The fused oxide is a conductor of electricity, but I have never been able to cast an anode of it which would withstand the disintegrating effect of the electrolyte. The action of the electric current upon an anode made of pure vanadic acid in an electrolyte of a water solution of the acid also slowly causes the formation of the yellow variety of the pentoxide, which form does not adhere to the crystalline mass.

The remaining oxides of vanadium are at present of no particular importance to the metallurgist, except it may be noted[5] that investigators have mistaken a regulus of the monoxide for the metal itself.[15]

Other Vanadium Salts.

Besides the oxides the following compounds of vanadium are known, and have been investigated: Chlorides, [5, 6, 7, 8, 9, 15]; oxychlorides, [6, 7, 8, 9, 15]; bromides and iodides, [9, 15]; fluorides, [10, 11, 12, 10]; sulphates, [14, 13]; sulphides, [15]; selenides and selenates, [17]; nitrides, [16]; nitrates, [16]; phosphates, [10]; arsenates, [10]; borates, [10]; silicates, [10]; silicides, [8]; carbide. The vanadates of sodium, potassium, calcium, ammonium, lead, mercury, copper, silver, barium, molybdenum, tungsten, uranium, manganese, nickel, cobalt and iron are well known, and need no reference.

I have investigated vanadates of aluminium, zinc and gold in addition to those mentioned above, as well as vanadyl oxalate and acetate. It is probable that vanadium forms stable salts with magnesium, platinum, osmium and iridium, and the rarer metals, although little has been published regarding these compounds. Vanadium also forms alloys with the common metals, and these alloys, containing only a small percentage of vanadium, have for the most part distinct properties, some of which are very valuable.

I shall discuss only the chemistry of those compounds of vanadium which are of importance in metallurgical operations of extraction or incident thereto.

Three chlorides of vanadium are known[18] (p. 709), two of which, VCl_3 and VCl_2, are formed from the third, VCl_4. The tetrachloride is formed by passing dry chlorine over metallic vanadium at a red heat[18] (p. 710) [5](p. 345).

Dammer mentions six oxychlorides of vanadium[18] (p. 710), three of which are said to be derived from the vanadium oxytrichloride. The statement is made that this compound, which has the formula $VOCl_3$, is prepared by heating a mixture of vanadium pentoxide and carbon in an atmosphere of dry hydrogen, allowing the mass to cool, and then heating in a stream of chlorine. Divanadyltetrachloride $(VO)_2Cl_4(H_2O)_5$ is formed by the action of hydrogen sulphide upon a hydrochloric acid solution of vanadium pentoxide with the separation of sulphur. The compound $2VO_2 . 4HCl . 3H_2O$ is said to result from heating vanadium pentoxide in a strong solution of hydrochloric acid, with the evolution of chlorine.

The action of concentrated hydrochloric acid upon vanadium pentoxide or upon a dried metallic vanadate, such as iron vanadate, results in the formation of vanadyl dichloride, $VOCl_2$, date, or upon vanadic acid, results in the formation of vanadyl dichloride $VOCl_2$ with an evolution of chlorine gas, according to the reaction,

$$2V_2O_5 + 12HCl = 4VOCl_2 + 6H_2O + Cl_2$$

The bromides and iodides of vanadium are analagous to the chlorides, and there are the corresponding oxysalts.

Melikoff and Kasanezky[19], and Ephriam[20] develop the following structural formula for a vanadyl ammonium fluoride which is of interest as showing the general character of all vanadium compounds which are in the highest state of oxidation:

$$\begin{matrix} NH_4-O- & & -Fl-Fl-NH_4 \\ NH_4-O- & V & -Fl \\ & & -Fl \end{matrix}$$

Vanadium pentasulphide is probably formed by the addition of ammonium or sodium sulphide to an alkaline solution of vana-

[4]Sir William Crookes, "Select Methods in Chemical Analysis." Ed. 1905, p. 186.

[5]*Zeit. f. anorg. Chem.*, 1905, p. 345. J. Koppel, R. Goldman u. A. Kaufmann. Verbindungen des vierwertigen Vanadins.

[6]*Zeit. f. anorg. Chem.*, 1902, p. 456. F. A. Gooch u. L. B. Stookey. Die Reduction der Vanadinsäure durch Chlorwasserstoffsäure.

[7]*Zeit. f. anorg. Chem.*, 1902, p. 66. F. Ephriam. Uber die Einwirkung von Chlorwasserstoffsäure auf Vanadinsäure.

[8]*Zeit. f. anorg. Chem.*, 1902, p. 71. F. Ephriam. Versuche zur Darstellung Chlorhaltiger Doppelverbindungen des Vanadinpentoxyds.

[9]*Zeit. f. anorg. Chem.*, 1904, p. 246. F. A. Gooch u. R. W. Curtis. Die Einwirkung von Halogenwasserstoffsäuren auf Vanadinsäure.

[10]*Zeit. f. anorg. Chem.*, 1901, p. 242. P. Melikoff u. P. Kasanezky. Fluorvanadinverbindungen.

[11]*Zeit. f. anorg. Chem.*, 1902, p. 80. F. Ephriam. Notiz über die Konstitution der Vanadindoppelfluorid.

[12]*Zeit. f. anorg. Chem.*, 1904, p. 442. P. Melikoff u. P. Kasanezky. Uber die Konstitution der Fluorvanadin Verbindungen.

[13]*Zeit. f. anorg. Chem.*, 1905, p. 352. J. Koppel u. A. Kaufmann. Versuche zur Darstellung von Vanadinmetall und einigen Vanadinverbindungen.

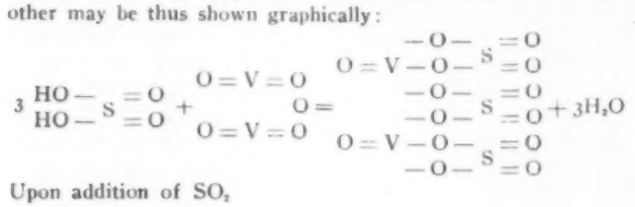
[14]*Zeit. f. anorg. Chem.*, 1902, p. 55. A. Piccini u. L. Marino. Uber einige Vanadinverbindungen von der Form VX_2.

[15]*Zeit. f. anorg. Chem.*, 1904, p. 152. L. Marino. Uber das electromotorische Verhalten des Vanadins.

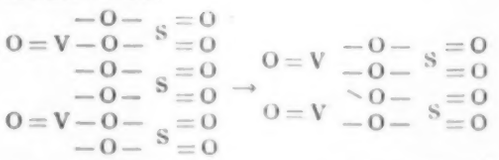
[16]Dammer, "Handbuch der anorganische Chemie."

dium pentoxide, out of which solution a mineral acid will precipitate the vanadium sulphide incompletely as a brownish colloidal mass, a portion of which can be collected upon a filter. Dammer[15] (p. 716) says that none of the sulphides are formed by the addition of H_2S to a solution of a vanadium salt, but that the precipitate formed by the addition of hydrogen sulphide to an alkaline solution is but a mixture of an oxide (he does not say which) and elementary sulphur. Just why an oxide should precipitate out under such conditions is not clear, and so far as I know the subject awaits investigation.

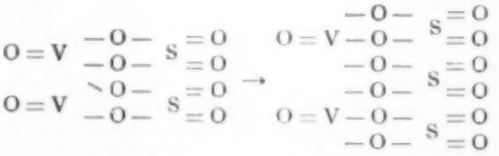
The vanadyl sulphates are important, there being five mentioned by Dammer[15] (p. 716). The usual reaction by means of which the vanadyl sulphates are converted from one to the other may be thus shown graphically:



Upon addition of SO_2



Containing V^3 and V^5 in equal parts. Upon addition of H_2O_2 or $KMnO_4$



According to the reaction:

$$5(VO)_2(SO_4)_3 + 2KMnO_4 + 8H_2SO_4 = 5(VO)_2(SO_4)_2 + K_2SO_4 + 2MnSO_4 + 8H_2O$$

Vanadium unites with silica chiefly in combination with other elements, but there is at least one compound containing only vanadium, silicon and oxygen. I have caused silica and vanadium to unite by mixing finely divided quartz with red vanadic acid, then fusing the mixture. The resulting mass is brown, or black, due to the uncombined vanadic acid. Upon grinding this fused mass and washing it with weak sodium carbonate solution, there remains a white residue which consists principally of quartz, but which contains vanadium. In using impure silica (free from vanadium) I have obtained upon washing with alkali a green residue which could not be distinguished with the eye from the original vanadiferous sandstone. This residue contained vanadium which was with difficulty extracted with nitric acid. I have also prepared a compound of vanadium and silica by treating an alkaline solution of sodium silicate with red vanadic acid added in the powdered form. The jelly-like precipitate which resulted was poured into a filter and washed until the wash water (containing a little alkali) showed no trace of vanadium. The product was white when dried, but various samples were not of constant composition. The precipitate when wet was distinguished from pure silicic acid by the readiness with which it could be washed on the filter. The same material probably results upon the addition of acetic acid to a solution of sodium vanadate containing also sodium silicate.

[17] *Zeit. f. anorg. Chem.*, 1907, p. 393. W. Prandtl u. F. Lustig. Uber Vanadinselenverbindungen.
[18] *Zeit. f. anorg. Chemie*, 1902, p. 174. F. A. Gooche u. R. D. Gilbert. Die Fällung des Ammonium Vanadats durch NH_4Cl.
[19] *Zeit. f. anorg. Chemie*, 1902, p. 181. A. Rosenheim. Die Fällung von Ammonium Vanadat durch Chlorammonium.
[20] A. H. Low, "Technical Methods of Ore Analysis," Ed. 1905, p. 205.
[21] Brearly, "The Analytical Chemistry of Uranium." Ed. 1903, p. 31.
[22] *Zeit. f. anorg. Chem.*, 1907, p. 368. T. F. Rutter. Beiträge zur Kenntnis der Vanadinverbindungen.

Alkaline Vanadates.

The vanadates of sodium are of importance because of their formation in all of the proposed methods of roasting crude ore to effect extraction. There are nine of these sodium vanadates, of which Dammer[16] (p. 721) gives seven. Of these the metavanadate, $NaVO_3$, and the pyrovanadate, $Na_4V_2O_7$, are easily soluble in cold water, the remainder are soluble with more or less difficulty, the dihypovanadate and the hexavanadate being almost insoluble in water, either hot or cold. All of the simple sodium vanadium compounds are decomposed by mineral acids, with the formation of vanadyl salts, and, if the solutions are highly concentrated, with separation of vanadic acid.

The vanadates of potassium are for the most part analagous to the vanadates of sodium, but they are not of metallurgical importance because of the relatively great cost of the potassium salts from which they could be made.

There is a considerable number of compounds of vanadium, sodium and sulphur which I have prepared in large quantities but have not yet had the time to analyze. One compound in particular occurs in the form of bright red crystals, very difficultly soluble in water and slowly soluble in mineral acids. It carries about 78 per cent V_2O_5, the remainder being sodium, sulphur and oxygen.

An acid added to a water solution of sodium vanadate will form vanadyl salts if the solution is dilute, and will form vanadic acid if the solution is concentrated.

If to a concentrated solution of sodium vanadate, which has been electrolyzed in the anode compartment of a porous cell, ammonia is added, a complex sodium ammonium vanadate is precipitated, and may be readily filtered and washed. The salt is crystalline and of a bright red color. The precipitation is not quantitative, but seems to be an instance of the formation of a salt by the addition of a relatively weak to a strong base.

Dammer gives seven distinct ammonium vanadates, analogous to the vanadates of sodium, of which the metavanadate is by far the most important, owing to its insolubility in ammonium chloride.[16-18-19] Ammonium vanadates are decomposed with alkalis and with mineral acids, in the latter case with partial separation of vanadic acid if the operation is performed with care and with this object in view. Ammonium metavanadate is formed by the addition of a large quantity of ammonium chloride to a concentrated neutral solution of sodium vanadate. It is decomposed on heating, leaving a residue of vanadium pentoxide in a state of purity corresponding to the purity of the ammonium compound. By repeated resolution of ammonium metavanadate in alkali, and reprecipitation with pure ammonium chloride, practically pure vanadic acid may be obtained by heating the final product. I have never been able to obtain a quantitative precipitation, although this is possible.[8-22-23] The salt is pure white, crystalline, filters readily, and may be washed with a moderate amount of hot water without appreciable loss if the water contains a small amount of ammonium chloride.

Metallic Vanadates.

Of the metallic vanadates it may be said in general that they are formed by the addition of a soluble salt of a metal to a neutral solution of sodium or potassium vanadate. They all carry much water when pressed as dry as possible, and they are all very light when dried, but not ignited, with the exception of the vanadates of mercury, lead, gold and barium, all of which are slightly heavier in gravity than the others. They are all decomposed upon the addition of mineral acids, usually with the separation of vanadic acid, and are reprecipitated upon the addition of alkalis to neutrality. All of the metallic vanadates seem to be soluble in water to a greater or less extent, but for the most part they are insoluble in water containing a very small quantity of the precipitating agent. All of the metallic vanadates when precipitated chemically may be colloidal, and some of them are always so. This is especially true of the vanadates of iron, copper, zinc and aluminium. Those

least tending to form colloids are the vanadates of lead, mercury and calcium.

The vanadates of mercury, lead, copper and iron are fusible at a temperature of about 600° C., while the vanadates of aluminium, calcium, zinc and tungsten are infusible at temperatures obtained with a blast lamp under the conditions of my experiments. The fused vanadates of iron and copper were found to conduct electricity to a sufficient degree to warrant this property being taken into consideration in trying to solve the metallurgical problems. I am not at this time sure, however, that the pure compounds would act in this manner, although the melting has been conducted in such a manner as to preclude the possibility of a partial reduction of one or more of the elements to the metallic state, and the conductivity of the material was seemingly the same as that of vanadate fused in graphite crucibles.

The fused vanadates of iron and copper are very heavy, probably having a specific gravity equal to iron or thereabouts, and are extremely hard. In fact, a hack-saw is dulled after a few strokes across the ingot, and the best grade of steel drill becomes greatly worn during an attempt to obtain drillings. The ingots may be broken with a hammer perhaps a little more readily than hard quartz, and present on the interior a crystalline structure which is much harder than the outer layer, which does not appear to be crystalline. The crystals found in the vanadate of iron ingots are much larger than those found in the vanadate of copper.

Since the metallic vanadates are not fully discussed in any of the papers which I have read, I shall devote a small amount of space to describing briefly those which I have studied.

By far the most important metallic vanadate is the vanadate of iron. This is a general name given to the ortho, meta, pyro and tetra vanadates, as well as to a mixture of one or more of these with ferric or ferrous oxide. Vanadate of iron is prepared by the addition of a soluble salt of iron, usually the sulphate, to a solution of sodium vanadate. The precipitate may be either black, green or yellow. The precipitate is usually colloidal, and contains considerable vanadium combined with sodium, as well as iron, which salts are slightly soluble in water.

The salt when dried is in the form of lumps, which may easily be crushed to a light powder, which may be either green, yellow, brown or red, the olive green variety being of the highest grade, and probably either the tetra or the ortho vanadate, or a mixture of the two. The more nearly the vanadate approaches red in color the lower its grade, the V_2O_5 content in some of the very red material I have assayed carrying as low as 18 per cent V_2O_5.

Vanadate of iron is acted upon by mineral acids of all degrees of dilution, but less readily when calcined or fused. The separation of vanadic acid is slight, owing to the difficulty of dissolving the oxides of iron which seem always to be present. It is acted upon to some extent by strong solutions of the fixed alkalis with the formation of sodium or potassium vanadate and hydroxide of iron.

Vanadate of copper is of considerable importance in metallurgical operations. It may be formed chemically only under favorable conditions, and when so formed is usually very impure. The impurities are difficult to remove by the ordinary methods of washing, and further, the precipitation is usually not quantitative. If to a solution of sodium vanadate a solution of copper sulphate be added, copper vanadate will be precipitated. The precipitate is usually blue, although it may be yellow or even green, depending somewhat on the purity. The best method of producing vanadate of copper is the electrolytic method which will be described in detail later on.

Vanadate of copper is readily decomposed by the mineral acids, with the separation of vanadic acid. It is not easily decomposed with alkalis, being but slightly soluble in these reagents. The dried powder is light, and is very irritating to the nose and throat. It may be easily melted and cast into ingots. A graphite crucible is used for this work with fair success. The crucible is steadily attacked, and the average number of melts is about four per crucible. The copper vanadate is reduced, the copper appearing as metal and the vanadium as carbide and silicate. It is a remarkable fact that the reduced copper contains but a trace of vanadium, the greatest amount ever found being under 1 per cent. Copper containing even this small amount of vanadium appears to be considerably toughened, and is much harder under the file than ordinary copper.

Vanadate of mercury is of no especial importance except as to its use in a method of gravimetric analysis. It is formed by the addition of mercurous nitrate to a neutral solution of sodium vanadate. In the process of analysis the mercury is removed by distillation, but it is a difficult matter to remove it entirely except by prolonged heating at a fairly high temperature. It is readily decomposed by acids.

The vanadate of zinc may be made in at least two ways. It is formed when a solution of neutral zinc salt, such as the sulphate, is added to a neutral solution of an alkali vanadate. The precipitate is usually yellow, though sometimes almost white. It is flocculent and quite gelatinous. It may be washed with water by means of a suction pump, and when dried is of a light yellow color, and, as characteristic of all of the metallic vanadates, excepting lead and mercury, it has a very low specific gravity. Vanadate of zinc is readily decomposed by acids with a separation of vanadic acid. No commercial importance is attached to this method of vanadic acid production, since none of the reactions are clean, and the resulting product is of a low grade, being contaminated with zinc. Vanadate of zinc is formed also electrolytically from solutions of alkali vanadates.

Vanadate of silver is formed in a manner similar to vanadate of zinc. It is slightly more stable in the presence of mineral acids, but in general the reactions are of no especial importance.

Vanadate of gold is formed chiefly by precipitation from solutions of alkali vanadates with gold salts. It is formed with extreme difficulty by electrolytic methods.

Calcium also unites with vanadium to form a number of vanadates, of which the metavanadate, $Ca(VO_3)_2$, is the one usually met with. These vanadates are formed by the addition of soluble salts of calcium to solutions of alkali vanadates. The metavanadate is white and very porous when dry, but contains over 90 per cent of water when removed from the filter press in cakes. It is readily decomposed by mineral acids with a separation of vanadium pentoxide, and partially decomposed upon treatment with a strong solution of sodium carbonate. It is unaffected by the hottest flame of a Teclu burner, and it does not readily absorb moisture from the air. In cases where it is necessary to add lime to an electric furnace product for fluxing, the vanadate is a very convenient form. Vanadates of lime are found in mill solutions at all times unless tedious steps are taken to remove them, and are present in all residues.

Vanadate of aluminium is made both chemically and electrolytically, and has the same appearance when dried as vanadate of lime. It is soluble in the mineral acids with the separation of vanadic acid, and is decomposed to a slight extent by strong alkaline solutions. It is infusible at the ordinary temperatures, and is of no importance metallurgically. It is only interesting because it exists at all.

Vanadate of lead may be made both chemically and electrolytically from solutions of alkali vanadates. It is yellow and heavy. When fused it may be cast into ingots, but the fused mass is a non-conductor of electricity. Fused, it is of a dirty gray color, very hard and quite brittle. It is decomposed readily by all mineral acids without the formation of vanadic acid. When decomposed with sulphuric acid the resulting solution contains vanadyl sulphate, which may be evaporated and fairly pure vanadic acid recovered. The decomposition of

lead vanadate with sulphuric acid is not complete unless the acid is dilute and unless a large excess of the acid be used. The residue of lead sulphate is colored yellow and the last traces of vanadium can be removed only by solution of the lead sulphate in ammonium acetate, and making a second separation.

The precipitation of vanadate of lead chemically from solutions is not a satisfactory one, except under very favorable conditions. It is best accomplished in an acetic acid solution by the addition of lead acetate. Under these conditions the vanadate is precipitated quantitatively, and may be readily filtered and washed. The best solvent for vanadate of lead is nitric acid, in which both the vanadium and lead go into solution. The lead may be removed by any one of a number of methods, and the resulting vanadium solution is of a high degree of purity. I shall deal with the electrolytic production of vanadate of lead later on.

Necessity for Uniform Conditions in Work with Vanadium.

Investigators have stated with undoubted truth that vanadium, when it cannot unite with any other group to form salts, will unite with itself, with the formation of vanadyl vanadates.

The deportment of the various vanadates and vanadium compounds with reagents is at first very confusing, because a very slight change of conditions will cause the formation of salts in different degrees of oxidation, and the chemist may get a precipitation when he expected a solution, or vice versa, or what is still more troublesome, a quantitative result in neither instance. It has been my observation and experience that this is the rock upon which almost all metallurgical methods for vanadium treatment have been wrecked. One of the first requisites for a successful metallurgical method is therefore a means of keeping the vanadium in one or the other state of oxidation during a given operation, and if possible, to predetermine what the stage of oxidation shall be. Unless this is done there can be no uniformity of results and success will be extremely uncertain.

A standard work on technical analysis directs the chemist to dissolve the vanadium in ammonium carbonate to effect the separation of vanadium from iron, etc.[20] Another writer[21] advises "pouring the filtered HNO_3 solution of the mineral (ore) into sodium carbonate, which causes the vanadium to go down with the iron." Now the action of sodium, ammonium and potassium carbonates upon compounds of vanadium is practically the same under the conditions of the directions quoted above. But if vanadium solutions are concentrated, or contain much iron or alumina, or are in a reduced (V_2O_4) condition, these reagents will cause a more or less complete precipitation, especially if the alkali is not added in large excess. Indeed, under the most favorable conditions of dilution, oxidation, etc., some of the vanadium may remain with the precipitate of iron and alumina; but only when the molecular quantity of iron or alumina or lime exceeds that of the vanadium can the vanadium be precipitated quantitatively, and then only when the solution is barely neutral.

The statement is met with among those who have not carefully studied the reactions of vanadium that the element is elusive, and does not act consistently in similar cases. This statement is, of course, not true, but the foundation for it lies in the fact that conditions which would in no wise affect other and more common elements cause the vanadium to change its state of oxidation, to become either acid or basic, or both, or cause it to eliminate itself from the field of action by precipitation. Such are the slight changes in oxidation which will cause a solution containing iron, vanadium, sodium and sulphuric acid to precipitate when exposed to the air. This reaction is of extreme importance, as it has a direct bearing upon the extraction of vanadium from the ores, and will be discussed later.

Ionization of Vanadium Compounds.

Dilute solutions of vanadic salts are yellowish in acid when dilute and blood red when concentrated. Upon dilution the red color disappears, and the yellowish returns. This may be explained by the fact of the great degree of ionization in dilute solution, and the solution of the compound itself when concentrated.

The action of hydrogen peroxide upon acid solutions of vanadium salts is interesting, the reaction being generally considered as one of oxidation only. Rutter[22] suggests that the action of H_2O_2 causes the formation of pervanadic acid, which is not really a compound of vanadium in a higher state of oxidation, but simply an addition compound.

The phenomenon may be explained by considering the action of H_2O_2 to cause hydrolysis as indicated by the formula:

$$\begin{array}{l} \\ -O-N=O \\ -O-N=O \\ O=V\ -O-N=O \\ -O-N=O \\ -O-N=O \\ -O-N=O \\ O=V\ -O-N=O \\ -O-N=O \end{array} \quad + x\,H_2O_2 \quad \begin{array}{l} O=V=O \\ \quad -O \\ O=V=O \end{array} \quad + 6HNO_3$$

(yellow) (blood red)

The fact that concentrated acid (sulphuric and nitric) solutions of vanadium are red, in which case we know the vanadium to be largely in solution as V_2O_5 because of the readiness of the pentoxide to separate out, leads me to favor the ionization theory rather than the theory of the formation of an addition compound. It has also been observed that after a portion of the vanadic acid has been precipitated from a concentrated acid solution the red color is in part discharged. Concentrated hydrochloric acid solutions of vanadium do not illustrate the red color because of the reduction which takes place.[6-7-8-9]

(To be continued.)

An Improvement in Hot-Blast Copper and Lead Smelting.

BY CLEMENT H. MACE.

A blast furnace embodying several novel features has been perfected after some years of experimenting by the inventor, Allen R. Partridge, and a small plant has just been built under

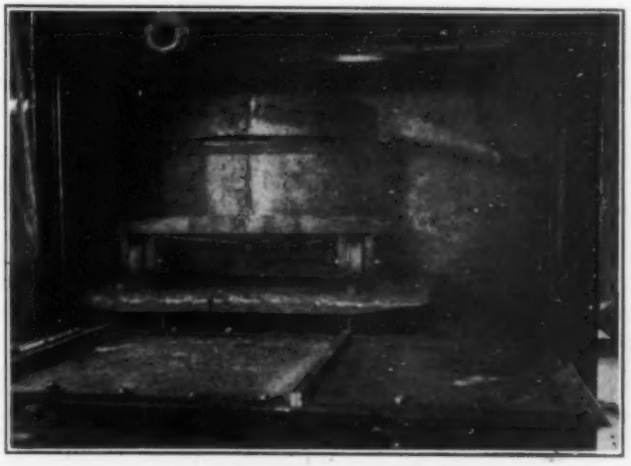


FIG. I.—CRUCIBLE CHAMBER.

the direction of the writer for the Alamos district, Sonora, Mexico.

There are several innovations from the standard lead or copper blast furnace, notable among which are the absence of

tuyeres and preheating the blast by molten slag. In place of a bustle pipe and tuyeres are a crucible chamber and a complete open space of 360° circumference and 2½ in. in height between the top of the crucible and the bottom of the bosh where the water jacket terminates.

In Fig. 1 the crucible is shown inside the crucible chamber on the jack plate, by means of which it is raised into position below the bosh. The slag overflows (it is never tapped) through

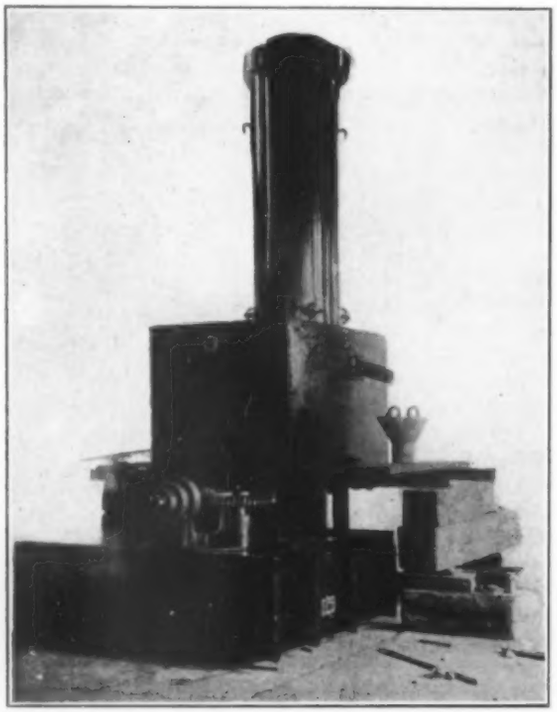


FIG. 2.—SMALL COPPER AND LEAD BLAST FURNACE.

the spout on the right, while the matte tap is on the outside of the crucible chamber (Fig. 2) and is made airtight by a clay packing on the spout block. The slag is carried in small, shallow cars that travel slowly beneath the slag tap and thence around the circumference of the slag box, finally dumping automatically into a boot in a water seal below. Here a perforated bucket elevator raises it to a car to be run to the dump.

The blast enters the slag box above the point where the cars discharge, passes over the cars of molten, inert slag (i. e., inert as far as the oxygen of the air blast is concerned) and up into the crucible chamber, where it then receives the radiation from the crucible itself. It thus enters the smelting zone at a high temperature. Consequently the molten material is not robbed of any heat to raise the temperature of the air to its level, so that the blast performs its duty of oxidation without the constant fluctuation of temperature inevitable at the tuyeres of the standard smelter due to the chilling effect of cold blast on the slag. It is perfectly evident then that a certain number of heat units are constantly being put back into the furnace and as a result less fuel is required to operate it. This method of heating the blast seems certainly preferable to others now in vogue; as, for example, an external stove, which requires extraneous fuel and has a short life, due to continual contraction and expansion, or to the system of utilizing the radiation from flue gases, which does not heat the blast sufficiently.

This furnace has demonstrated the fact that the introduction of the blast through tuyeres at a high pressure is erroneous. This applies to a reducing smelt as well as an oxidizing one, and is especially evident in "pyritic smelting." In order to smelt correctly it is essential that the blast enter the zone of fusion at a rate of speed consistent with maintaining an active combustion, but the excessive blast-forge effect is wrong. It causes heavy flue dust, an increase in the height of the fusion zone, and losses of metals by volatilization.

All of these disadvantages are overcome in this furnace and a pressure of from 4 oz. in small furnaces to 12 oz. in large ones is maintained. In other words the actual *air inlet* is increased and an enormous volume of air enters the furnace at a low pressure and velocity, enabling the oxygen to react thoroughly on the sulphides, thereby serving the dual purpose of oxidizing their iron content for fluxing the silica and utilizing the heat of the sulphur. Thus no preliminary roasting is necessary and high-grade mattes, or even black and blister copper, are obtained.

On a run made at the Denver experimental plant, under the supervision of the writer, on ore from Central City, Col., the following results were obtained:

	Ore.	Slag.	Matte.
SiO_2	71.2 per cent.	46.5 per cent.	
FeO	9.8 " "	29.0 " "	
CaO	0.8 " "	15.0 " "	
Al_2O_3	4.5 " "		
Cu	2.73 " "	Tr	14.4 per cent.
Au	0.29 oz.	nil	1.51 oz.
Ag	2.6 "	Tr	11.98 "

Coke used was 8 per cent of gross charge.

The ore was fluxed with metallic iron, which accounts for the low rate of concentration.

Iron matte smelting can be satisfactorily practiced when the

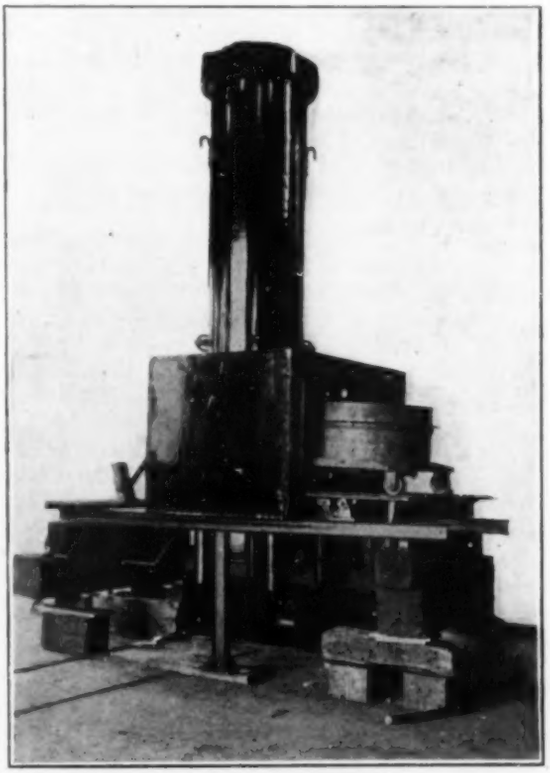


FIG. 3.—SMALL COPPER AND LEAD BLAST FURNACE.

copper content of the ore is very low or nil, but at least 15 per cent FeS_2 should be present and 0.25 per cent Cu to give a clean slag. The smelter is equally adaptable to lead ores, the only change in construction being a lead well connected to the crucible, and for furnaces of a greater capacity than 60 tons a forehearth is used.

The power required for the smaller sizes is as follows:

5	tons	per	24	hours	.....	3 hp.
15	"	"	"	"	.....	6 "
30	"	"	"	"	.....	12 "
50	"	"	"	"	.....	20 "

These sizes can be built in small sections, if necessary, for mule-back transportation, and riveted at the mine.

The ore shaft is water-jacketed with hand holes at the bottom to clean out sediment and scale and thus prevent the bottom plate from burning out. The water jacket discharges into the slag pit and the water seal thus formed prevents the blast escaping as the cars dump. Mica peep holes in the top of the crucible chamber allow the furnace man to watch the operation of his furnace. The only brick work necessary is that for the crucible lining.

The operations of "blowing in" and "blowing out" are very simple and rapid. By turning a crank, which operates a set of miter and spur gears, the four jack screws raise or lower the jack plate on which the crucible rests on a 27-in. gage track. The crucible chamber door has on its inside face a track that is in juxtaposition to that of the jack plate when the door is open and horizontal. This facilitates change of crucibles, as an old crucible may thus be run out on a T-rail track extending to a turn sheet outside of the building and an auxiliary crucible, previously heated, be run in, the door closed, and the crucible raised into position below the bosh. At the "blow in" the slag will be running within 10 minutes of turning on the blast.

The furnace shown in the cuts has an ore shaft 22 in. in diameter and is boshed at the bottom to 18 in. It has a rated capacity of 15 tons per 24 hours and is intended to meet the wants of the small operator. The standard type of smelter would surely "freeze" if made as small as this, for the introduction of cold air under a high pressure through small tuyeres has a tendency to chill a small furnace, and "muscular metallurgy" is often necessary. In fact, the smallest circular furnace carried in stock by the supply companies has a diameter at the tuyeres of 30 in. for copper smelting and 36 in. for lead smelting and these would have a capacity of only 20 tons to 30 tons. The open fusion zone and hot blast account for the free running and rapid driving of the Partridge furnace and it has a minimum smelting capacity of 8½ tons per square foot of furnace area for copper as against 6½ tons, the best result obtained by the standard smelter.

It is believed that this smelter will prove of great value to base metal mining in general and to small, inaccessible copper and lead properties in particular, such as those located in Central and South America, Mexico, Alaska and the western part of the United States.

Some Alloys for Permanent Magnets.

BY C. F. BURGESS AND JAMES ASTON.

In the investigation of electrolytic iron and its alloys, carried out at the University of Wisconsin under grant of the Carnegie Institution of Washington, tests were made of the magnetic quality of all samples. Among these many showed low permeabilities and high coercive forces, suggesting their utility for permanent magnets rather than as electromagnetic material. Upward of 100 samples were tested for this feature; all of the bars contained chromium, manganese, molybdenum, nickel and tungsten as binary alloys, or in various combinations together, or with other elements.

Much information regarding the best steels and the best treatment for permanent magnets has undoubtedly been accumulated by the manufacturers and users of such material, but is held as a trade secret, and published data along these lines is very meager. Again, such publications deal largely with the study of the critical points of the steels, with the view of determining the proper heat treatment to give them for the best service.

The tests here recorded are for the most part on carbon-free alloys. The results are mainly of value, therefore, as indicating the influence of the various elements in inducing the conditions necessary for a good permanent magnet. They were made upon an Esterline permeameter, an apparatus giving directly the flux density in the bar under test, for any impressed magnetizing force. It consists of a small dynamo, driven at predetermined constant speed. The test bar forms part of the magnetic circuit of the generator and can be subjected to magnetizing forces of varying degree. The flux density in the bar, being directly proportional to the voltage set up, can be read off on a properly calibrated millivoltmeter.

It is not necessary to make a complete magnetization curve in testing material for permanent magnets. The desirable feature is to have as high a residual magnetism as possible, or better, to have a high retentivity after some disturbing factor has tended to destroy the residual magnetism, together with a high coercive force to resist the tendency toward this loss of the magnetic flux. The method of procedure was as follows: The bar was inserted in the permeameter, the magnetizing force H was raised to the maximum of 200 and the density $B\ (MAX)$ was recorded. H was decreased to zero, and the retentivity $B\ (RET)$ noted; next the coercive force Hc necessary to reduce the retentivity to zero was determined. Similar readings were taken for the reverse magnetizing forces, and the average of these positive and negative readings of $B\ (MAX)$, $B\ (RET)$ and Hc for each bar was recorded for comparison. In fact, the whole operation is identical with the taking of a hysteresis loop in which only the few readings noted above were recorded.

The bars were tested as forged, and after water quenching at 1000° C. To prevent oxidation of the samples while heating them for quenching a barium chloride bath was used. The quenching at 1000° C. may not have brought out the best qualities of some of the bars, but it was deemed the best general treatment, since with so many bars under test it was impossible in this research to subject each sample to tests for its critical points and special conditions of heating.

In addition to the general test noted above all bars which showed a fair retentivity and a coercive force above $Hc = 30$ were subjected to a test for their stability, or their ability to hold residual magnetism after shock. For this test the bar was subjected to the maximum magnetizing force of $Hc = 200$, and the retentivity noted when this magnetizing force was reduced to zero. The bar was then removed from the machine, subjected to the disturbing factor, and the retentivity again recorded after reinserting the sample in the permeameter. The two means commonly employed in practice for seasoning permanent magnets were used, namely, jarring or rapping the bar and boiling. For the latter treatment each bar was inserted in a piece of wrought-iron pipe, which served as a shield to prevent the mutual influences of the magnetic fields and boiled in water for three hours.

A large number of comparative trials of the rapping and boiling treatments showed them to be practically equal in their effect, and because of the simplicity of the former it has been resorted to in the later tests. However, should the material be very brittle it would be safer in practice to use the boiling method.

As indicating the efficiency of the rapping treatment there are noted below the results of successive jarring upon the retentivity of a bar of composition 7 Cr 3W 2Mo.

$H\ (MAX) = 200.$
$B\ (MAX) = 11{,}100.$
$B\ (RET) = 7500.$
$B\ (RET) = 4900$ (removed and rapped 1).
$B\ (RET) = 4700$ " " " 2).
$B\ (RET) = 4700$ " " " 3).

As will be seen, but one vigorous rapping is necessary, since this reduces the residual magnetism to a value which is but little changed by further shock.

Only the general summary of the tests is given, since it was not deemed advisable in a paper of this character to include the complete tabulation of the data. In Table I are noted the data of the stability tests which, as mentioned previously, include only those materials best suited for permanent mag-

nets. The alloys are divided into five groups according to their content of chromium, manganese, molybdenum, nickel and tungsten, either alone with iron, or together with iron and any other elements except the above five; in addition, a sixth miscellaneous group includes materials of various compositions in which more than one of the above five metals are present.

The results as a whole showed a wide variation in the physical and magnetic hardness of the various bars, not only before quenching, but afterward as well. Also, it appeared that the effect of quenching was varied in its results, with either increase or decrease of coercive force, or with no material alteration. However, it appeared that where marked changes were the result it was generally with an increase of magnetic hardness.

Chromium.—In the chromium group, it appears that this element alone, even in percentages as high as 16.65, does not result in bars of high coercive force suitable for permanent magnets, even after quenching at 900° C. And the coercive force was no higher with large than with small chromium additions.

A large number of chrome-silicon alloys were tested, and in practically all cases the bars were relatively soft and of low coercive force in the forged condition. The coercive forces in all but two instances were well below 30, and the alloys were thus unsuited in this state for permanent magnets. In these two instances but little change resulted on quenching; but for the other bars quenching resulted in a marked increase of coercive force, and with the accompanying relatively high retentivities these materials were well adapted for permanent magnets. The best alloys seemed to have about a medium chromium content (5 per cent or 6 per cent) since then the retentivity was high and the coercive force great enough to insure stability.

The addition of vanadium also seemed to be very beneficial, although in this case the raw forged bars were especially hard, and the coercive forces were high initially, and suffered but little change in quenching. Here, again, the best results seem to be gained with medium chromium content because of the higher retentivities.

In a few instances carbon was present, together with vanadium and silicon. The materials showed hardening upon quenching, but the magnetic results were no better than where there was no carbon.

Summarizing, it may be said that the addition of chromium alone to carbonless iron will not result in a material suitable for permanent magnets, but that the further addition of 0.75 per cent to 1 per cent of silicon or 0.3 per cent to 0.5 per cent of carbon to 5 per cent or 6 per cent of chromium is highly beneficial and gives an alloy very suitable for this class of service.

Manganese.—According to the classification adopted, the only manganese alloys falling in this group were of a series in which manganese alone in varying amounts had been added to the iron. In the raw forged condition the coercive force increased, with a corresponding decrease of retentivity, for each increase of manganese content until at 10.41 per cent Mn the alloy was non-magnetic. A bar with 10 per cent of manganese had the high coercive force of 90.1, but the rententivity was too low for permanent magnets. The most satisfactory bars were two with 4.51 Mn and 6.0 Mn, where the retentivities were fairly high and the coercive forces great enough to insure stability.

But three bars were tested after quenching, and the result was a decrease of coercive force; a peculiarity corroborated by other tests upon this series.

Molybdenum.—Of the entire set of results for this series of alloys only one was magnetically hard in the forged condition; this was a bar of composition 8Mo, 0.3V, 0.6C, which had the high retentivity of 9200 and the fair coercive force of 39.0. In the quenched condition there was practically no change in any of the carbon-free alloys. But in four cases where carbon was present the quenching resulted in physical hardening of the bars and they became very fine materials for permanent magnets, with exceptionally high retentivities and very good coercive forces. In one case a bar with 4Mo and 0.6V attained a good coercive force; whether this was due to absorbed carbon is problematical. If the magnetic hardening took place without the aid of carbon it would indicate that the proper conditions of temperature, etc., might have the same effect on the other molybdenum alloys. This is merely conjecture, however, and the tabulated results showed no hardening effect of vanadium, such as was noted for the chromium alloys.

Nickel.—In the binary alloys of nickel and iron, increase of nickel caused a decrease of retentivity and increase of coercive force until at about 26 per cent of nickel the material was non-magnetic. Beyond this percentage the magnetic properties were restored, the alloys being of rather high permeability and low coercive force. Quenching at 900° C. invariably greatly decreased the coercive forces, so that none of the binary nickel alloys can be considered suitable for permanent magnets in the quenched condition, either because of low coercive force, or because of low retentivity where the coercive force is ample.

The addition of vanadium resulted in increased magnetic hardness, which was again increased by quenching. The most satisfactory bar was one with 10 Ni, 0.6 V, which in the forged condition had a retentivity of 9900 and a coercive force of 32.6; both values were slightly lowered by quenching. One bar with 10 Ni, 2 Si, 1 C had a fair retentivity of 6800 with a high coercive force of 62.0 in the forged condition, but after quenching it became almost non-magnetic.

The nickel alloys as a whole, therefore, cannot be regarded as satisfactory materials for permanent magnets. In this case, as noted for chromium, the vanadium seems to act as a hardening agent.

Tungsten.—Seven binary alloys of tungsten and iron were recorded, with a tungsten content varying from 0.93 per cent to 23.87 per cent. In no instance was the coercive force high enough to make them suitable for permanent magnets; and quenching had but little effect, usually resulting in a decrease of coercive force. For the two alloys where there was an addition of vanadium, one bar with 4 W 0.4 V became, after quenching, an exceptionally good permanent magnet, with a retentivity of 11,300 and coercive force of 43.8; on the other hand, another sample with 7 W, 0.3 V experienced but slight increase of hardness. A bar with 5 W, 0.5 C became noticeably harder after quenching, with increase of retentivity and coercive force to the high values of 11,700 and 59.0. Four other bars were very good for permanent magnets as forged, and there was a marked hardening due to quenching.

Summarizing, the binary alloys of tungsten and iron are not satisfactory for permanent magnets. The addition of vanadium may be beneficial, and the addition of carbon certainly is. There were in this series some highly satisfactory alloys; of those worthy of particular mention, however, all but one contained carbon.

Miscellaneous.—This miscellaneous group included such a large number of combinations of these several self-hardening alloys that a systematic discussion of the results is difficult. Of the 51 bars here grouped, almost two-thirds were good materials for permanent magnets. The effect of quenching was somewhat varied; in 13 instances there was a considerable increase of coercive force, in 30 cases but little change and in six instances a marked decrease. In four cases bars which were originally poor became of fair quality after quenching, and in numerous instances the quality of fair materials was greatly improved; in no case, however, has quenching resulted in a bar of fair quality becoming poor material for permanent magnets, although at times the quality was better before quenching than after.

One of the most striking features noted was the detrimental effect of the additions of nickel with its tendency toward non-magnetic alloys. In all cases where the material was classed

as non-magnetic, or where it was worthless for permanent magnets because of very low retentivity, there was a considerable amount of nickel present. Of the 17 bars of poor quality for permanent magnets, 12 had nickel, and of the five bars without nickel the quality was poor not because of a tendency toward being non-magnetic, but because the coercive force was too low. Several bars of nickel with chromium, with molybdenum, or with tungsten, had fair retentivities together with high coercive forces.

The various combinations of chromium with molybdenum or tungsten, either alone or with additions of silicon, vanadium and carbon, gave generally satisfactory results. In fact, it is difficult to point out any preferable combinations of these various self-hardening elements beyond the caution to keep nickel out of the alloy.

Of particular quality may be mentioned the following:

Bar.	*Composition.*	*B (Ret.)*	*He.*	
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	11,350	55.8	As forged.
136V	10 Ni, 5 Mo	10,150	36.7	As forged.
150C	2 Cr, 10 Mo, 0.5 C, 0.3 V	10,500	78.0	Quenched.
155Y	10 Ni, 5 W	10,300	35.2	Quenched.
156A	1 Mn, 2 Mo	10,500	39.6	Quenched.
155P	1 Mn, 10 Mo, 0.5 C	10,750	58.4	Quenched.

H(Max) = 200 PERMANENT MAGNETS—STABILITY TABLE I.

Bar.	*As forged*			*Quenched*		
Chromium. Composition.	*B(Ret).*	*B(Ret).*	*H(c).*	*B(Ret).*	*B(Ret).*	*H(c).*
133E 4 Cr, 0.84 Si	15050		14.9	9000	3500	41.5
133D 5 Cr, 0.84 Si	7500	3300	41.0	9500	4800	41.8
133B 5 Cr, 1.4 Si	15700		16.2	8800	3900	46.0
160J 5 Cr, 0.3 V	7400	3400	53.0	9000	2700	42.2
160H 5 Cr, 0.5 V	9000	3600	50.5	9100	4200	45.0
160K 5 Cr, 1 V	8400	4100	48.1	9200	4900	49.0
133L 6 Cr, 0.4 Si	15650		14.9	9600	3700	49.0
133K 6 Cr, 0.84 Si	15300		13.6	9400	5200	52.3
147Z 6 Cr, 1.3 Si	9200		16.2	8800	3400	50.0
133F 6 Cr, 1.68 Si	14300		17.9	8300	3800	50.0
133M 8 Cr, 1 Si	12550		16.2	8700	2400	48.5
111S 10 Cr, 2 Si	10000		19.2	7100	3600	54.0
155K 10 Cr, 2 Si, 0.3 V, 0.6 C	11200	2300	30.5	3000	1600	69.0
137L 10 Cr, 2 Si, 2 Ti	10850		16.4	7500	4200	62.0
122E 10 Cr, 3 Si	10800		12.3	7400	3500	47.8
160I 10 Cr, 0.3 V	7500	4000	72.5	7800	3500	58.0
155L 10 Cr, 0.3 V, 0.65 C	4600	3100	84.6	7200		91.9
160G 10 Cr, 0.5 V	7000	3600	65.0			
163O 10 Cr, 1 V	7600	3500	62.8	8400	4500	54.5
141C 10.3 Cr, 1 V, 2 Si, 0.3 C	9500		20.9	8000	2600	38.0
133N 13 Cr, 1.8 Si	12750		16.7	7100	4000	62.8
136H 17 Cr, 1.5 Si	2100	1100	89.3	4900	3700	85.0
Manganese						
170D 6 Mn	8500	4000	50.0			
Molybdenum.						
160E 4 Mo, 0.6 V	12950		15.3	10400	2600	33.3
156E 8 Mo, 0.3 V, 0.6 C	9900	3500	37.8	12000	5000	52.2
160C 10 Mo, 0.3 V, 0.4 C	13700		23.7	10400	4900	55.8
163H 10 Mo, 1 V, 0.4 C	10850		25.1	9400	5000	53.5
141F 10 Mo, 1.2 C				10100	4000	37.0
Nickel.						
160M 10 Ni, 0.6 V	9700	3400	30.8	9700	4000	27.2
146P 10 Ni, 2 Si, 1 C	6700	3200	58.7	1000	900	58.0
Tungsten.						
160P 4 W, 0.4 V	8250		12.4	11300	3300	37.5
163J 4 Wo, 0.4 V, 0.5 C	11200	3000	34.8	8800	4100	67.5
163D 4 W, 0.4 V, 0.6 C	10400	2900	33.0	9200	4600	68.2
163K 4 W, 0.4 V, 0.8 C	8900	2800	32.5	6600	4400	70.6
155X 5 W, 0.5 C	10350		24.5	12000	5200	52.5
WIC 6 W + C(?)	12500	2700	28.0			
163E 7 W, 0.3 V, 0.6 C	10900	3800	38.0	9600	5600	81.0
Miscellaneous.						
150C 2 Cr, 10 Mo, 0.5 C, 0.3 V	6900	3600	58.0	10700	6300	77.2
155S 10 Cr, 5 Mo	8700	4000	59.0	7300	3600	70.5
155U 2 Cr, 10 W	10100		23.4	9500	3900	49.0
156F 3 Cr, 8 W	9400		24.1	9900	4700	47.5
163A 5 Cr, 2 W, 0.6 C	6900	3800	61.0	7300	2500	55.0
163G 5 Cr, 5 W, 0.6 C	3400	1700	68.5	6400	4200	85.0
163C 10 Cr, 2 W	7400	4200	71.7	7500	4200	62.0
156K 10 Cr, 2 W, 2 Si	5800	3200	76.6			
156J 10 Cr, 2 W, 0.6 C	4900	3000	89.9	7100	4800	95.0
156H 10 Cr, 2 W, 0.3 V	6900	3700	66.0	6900	3800	61.4
156I 10 Cr, 2 W, 0.3 V, 0.6 C	4300	2500	82.7	7000	3900	91.0
155V 10 Cr, 10 W	5800	3300	80.5	6500	3600	72.7
150B 2 Cr, 10 W, 10 Mo, 0.3 V, 0.4 C	6100	3500	66.5	8400	3700	59.0
150D 2 Cr, 16 W, 5 Mo, 0.3 V, 0.5 C	5700	3600	64.0	8600	5000	63.5
163B 7 Cr, 3 W, 2 Mo	7300	4100	82.5	7400	2500	47.5
156G 10 Cr, 2 Mo, 2 W	7100	4100	78.5	6600	3800	65.8
156M 10 Cr, 2 W, 2 Mo, 0.6 C	4300	2500	82.0			
156P 10 Cr, 2 W, 2 Mo, 2 Si, 0.3 V	8500	1900	31.0	6800	2700	60.0
136M 5 Cr, 3 Mn, 4 Mo, 1 Si	10800	5300	64.7	5800	2900	72.2
140Q 5 Cr, 6 Mo, 10 Mn, 10 Ni, 1 C	10100	4000	38.2	7000	4400	85.3
144Z 2 Cr, 10 Ni	8100	3200	36.9	8200	3300	29.0
154E 5 Cr, 10 Ni	9100	4000	57.7	5400	2900	47.0
154F 7 Cr, 10 Ni	4700	2500	64.0	2000	1100	55.7
154A 10 Cr, 8 Ni	1600		80.8	800	300	73.5
136V 10 Ni, 5 Mo	9600	2300	36.0	8600	3600	32.0
146Q 10 Ni, 10 Mo, 1 C	5500	3000	69.7	5500	2800	70.4
146S 10 Ni, 10 Mo, 0.5 C	8900	4900	61.0	8400	4000	52.0
152Z 10 Ni, 5 W	8400	3100	36.0	8600	2800	29.7
155Y 10 Ni, 5 W	9050		17.0	10900	4600	38.0
155AB 15 Ni, 5 W	8300	3000	38.5	8000	2300	37.8
156A 1 Mn, 2 Mo	9450		27.3	10600	4600	38.5
155P 1 Mn, 10 Mo, 0.5 C	6900	3600	61.0	11200	5400	57.5
156C 2 Mn, 5 Mo, 0.6 C	9500	5500	57.2	8400	4700	68.0

Strange to say, two of these alloys have a high nickel content, but they may be put down as exceptions to the general rule. The values given above indicate a very high standard of material for permanent magnets.

Summarizing, it may be deduced from the above tests that chromium, manganese, molybdenum and tungsten are the important additions in the manufacture of steels for permanent magnets, but the presence of some third element is necessary. While carbon is beneficial, it is possible to obtain highly satisfactory materials either by various combinations of the above four elements or by suitable additions of silicon or vanadium, the latter, in particular, being very desirable. Also, suitable materials may be made which will require quenching, although in this case carbon is most beneficial, or, if desired, equally satisfactory results may be obtained without.

Stability.—As mentioned previously in the general discussion of the tests for permanent magnets, a better indication of the value of material for this service is obtained by a stability test which gives the residual magnetism after shock. In our tests the rapping method was used, the details being given above. Those bars were tested which showed a coercive force above 30, either before or after quenching, this being considered the minimum value at which permanence might be expected.

The details are tabulated in Table I in the same order as used in the general discussion. The figures of interest are those for the retentivity, B (Ret), after rapping; they show the marked decrease of the residual magnetism due to shock, a decrease averaging about 50 per cent, but varying according to the coercive force of the material under test.

For purposes of comparison, and for illustrating the above points, the following tables have been prepared. For Table II

Table II.

Bar.	*Composition.*	*B(Ret).*	*H(c).*	
WIC	(6 W + C) (2)	13,450	30.0	As forged.
156E	8 Mo, 0.3 V, 0.6 C	11,750	53.7	Quenched.
155X	5 W, 0.5 C	11,700	59.0	Quenched.
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	11,350	55.8	As forged.
160P	4 W, 0.4 V	11,300	43.8	Quenched.
163J	4 W, 0.4 V, 0.5 C	11,050	38.0	Forged.
155P	1 Mn, 10 Mo, 0.5 C	10,750	58.4	Quenched.
155K	10 Cr, 2 Si, 0.3 V, 0.6 C	10,700	32.9	As forged.
163D	4 W, 0.4 V, 0.6 C	10,650	35.5	As forged.
150C	2 Cr, 10 Mo, 0.5 C, 0.3 V	10,500	78.0	Quenched.
156A	1 Mn, 2 Mo	10,500	39.6	Quenched.
163E	7 W, 0.3 V, 0.6 C	10,450	39.4	As forged.
160E	4 Mo, 0.6 V	10,400	35.0	Quenched.
160C	10 Mo, 0.3 V, 0.4 C	10,350	59.8	Quenched.
155Y	10 Ni, 5 W	10,300	35.2	Quenched.
136V	10 Ni, 5 Mo	10,150	36.7	As forged.
141F	10 Mo, 1.2 C	10,000	37.4	Quenched.

Table III.

Bar.	*Composition.*	*B (Ret) After Rapping.*	*Hc.*	
150C	2 Cr, 10 Mo, 0.3 V, 0.5 C	6300	77.2	Quenched.
163E	7 W, 0.3 V, 0.6 C	5600	81.0	Quenched.
156C	2 Mn, 5 Mo, 0.6 C	5500	57.2	As forged.
155P	1 Mn, 10 Mo, 0.5 C	5400	57.5	Quenched.
136M	5 Cr, 3 Mn, 4 Mo, 1 Si	5300	64.7	As forged.
155X	5 W, 0.5 C	5200	52.5	Quenched.
133K	6 Cr, 0.84 Si	5200	52.3	Quenched.
150D	2 Cr, 16 W, 5 Mo, 0.3 V, 0.5 C	5000	63.5	Quenched.
163H	10 Mo, 1 V, 0.4 C	5000	53.5	Quenched.
156E	8 Mo, 0.3 V, 0.6 C	5000	52.2	Quenched.

the data have been taken from general tests and there are listed those bars having a retentivity above B(Ret) = 10,000 and a coercive force exceeding H(c) = 30.0, the order being accord-

ing to the values of B(Ret). In Table III the selections have been from stability Table I, and are ranked on the basis of the values of B (Ret) after rapping.

As will be seen by comparison of the two tables there are bars listed in Table III which do not appear in Table II. Also the order of rank is not the same for those materials which are noted in both tables. It will be noted, likewise, that some of the alloys which have high rank in Table II are low down in Table III, or do not appear at all, because the coercive forces are not great enough to retain a high residual magnetism after shock. All materials listed in Table III have high coercive forces, above Hc = 50, and it would appear that for permanent magnets to give good service it should reach this value.

As a result of these tests it appears that for the investigation of materials for permanent magnets this stability test gives a very satisfactory indication of the quality. In addition, of course, it would be necessary to test the ageing qualities, or the ability to retain this residual magnetism for long periods of time, and under working conditions.

Northern Chemical Engineering Laboratories,
Madison, Wis.

Temperature Drop, a True Measure of Thermal Resistance.

By Carl Hering.

In the calculations of the heat losses through the walls of furnaces, and in fact in most calculations of flows of heat, it has been the custom, dictated perhaps more by usage than by reason, to use the thermal conductivity of the material as the basis of the calculations, instead of the resistivity, although the latter is the more general practice in electrical calculations. There seems to be no good reason, however, for making this difference; on the contrary, it would probably be much better to follow electrical practice. The writer has repeatedly recommended making the change, and has often used thermal resistivities to advantage in place of conductivities. In an article in this journal, January, 1909, page 11, he showed that for the very cases occurring in electrical furnaces in which the walls are made of layers of different materials, it is much simpler to use resistances than conductances.

Besides the difficulty in overcoming what might be called mental inertia—that is, changing our accustomed methods—the chief objection to using thermal resistivities seems to be the lack of a named and easily understood unit for representing those quantities numerically. There is a natural and excusable hesitancy to use, in important calculations, units or quantities with which we are not familiar and about which there is a mistiness or an apparent mystery. Just as we do not like to purchase the proverbial cat in the bag, so we do not like to deal with units when we do not know what they mean or cannot refer to them by name.

Some time ago the writer showed that thermal conductivities may be correctly and rationally expressed and measured in terms of watts for either an inch or a centimeter cube and for centigrade degrees. Thus the thermal conductivity of graphite is about 3 watts for an inch cube; that of copper is about 10 watts, etc. Moreover, for electric furnace calculations in which the production of heat energy is naturally expressed in watts that unit is by far the simplest and most convenient one to use.

The physical significance of this unit is that, if the difference of temperature of 1 deg. C. be maintained between the two opposite faces of an inch (or centimeter) cube of graphite, there would be produced a flow of heat through it equal to 3 watts (assuming that there is no lateral loss); that is, the amount of heat flowing out of the cooler end would be equal to that set free by a coil of wire consuming 3 watts and setting it free as heat. Hence for thermal conductivities we now have a very convenient, simple, rational and easily understood unit with which we are already familiar, and one which has the additional advantage that it was born with a familiar name; hence, we need not even learn a new name. It was not necessary to create it, it required only to discover it.

The purpose of the present article is to show that there also exists an equally simple, convenient, rational and easily understood unit of thermal resistivity, which like its mate also required merely to be discovered and not created, and already had a name, being a well-known quantity with which we are all familiar, namely, a drop in temperature.

The drop of temperature in degrees can be shown to be a true measure of thermal resistivities, just as watts are a true measure of thermal conductivities. Moreover, these two units are the correct mates of each other, because the reciprocal of the numerical value of a conductivity expressed in watts is equal to the corresponding numerical value of the resistivity of that material expressed in degrees, and vice versa.

There is, therefore, no longer any good reason why we should not hereafter use resistivities instead of conductivities in thermal calculations, and there are often good reasons why we should do so. Tables of constants may also now be given quite as well in resistivity units as in conductivity units.

The physical meaning of this new unit of thermal resistivity is that if its value is, say, n°C., it means that it requires a difference or drop of temperature of n°C. between the two opposite faces of a cubic inch (or centimeter) to cause a flow of 1 watt of heat.

If, for instance, the thermal conductivity of graphite is 3 watts in centigrade-degree and cubic-inch units (or 1.18 in. cb. cm. units), its thermal resistivity will be 0.333° C. in cubic-inch units for 1 watt flow (or 0.848° C. in cb. cm. units), the two values being the numerical reciprocals of each other, as they should be in any rational system.

The electrical analogy would be to express electrical resistivities in terms of the difference of potential in volts required to cause 1 amp to flow through a cube of 1 in. This could be used quite as well as the more usual ohms for 1-in. cube. The higher the resistivity the higher this number of volts or degrees, and in exact proportion.

The watt and the degree are, therefore, convenient, simple and rational units for expressing and measuring thermal conductivities and resistivities respectively. But there are still wanting similarly convenient and simple units for expressing and measuring thermal conductances and resistances. The differences between these quantities are unfortunately not always clearly understood. Conductivities and resistivities are specific quantities; like specific gravity, specific heat, etc., they refer to some unit quantity of the material and are, therefore, physical constants, or properties of the materials, and are independent of the actual size or shape of a piece of the material. Conductances and resistances, on the other hand, refer to actual pieces of the materials of a given size and shape; they represent the amounts of these properties possessed by the pieces of the given sizes and shapes, and they are, therefore, dependent on the size and shape. Just as the total weight of a body depends on its size, while its specific gravity (the weight of a unit quantity in terms of water) is independent of its size, so the conductances and resistances depend on the actual size and shape while the conductivities and resistivities (or specific conductances and specific resistances) are independent of the size or shape.

The electrical resistance of a given piece of material is measured in ohms, while the electrical resistivity of that material is measured in the ohms which 1-in. cube has. In a precisely analogous way it is quite proper to measure a thermal resistance in degrees of temperature, when a thermal resistivity of the material can be measured in degrees for 1-in. cube. Hence, the total thermal resistance of a given furnace wall or a whole furnace may also be correctly and rationally expressed and measured in temperature degrees, strange as it may at first seem.

The meaning of such a thermal resistance expressed in degrees is simply that that is the number of degrees drop of temperature which would cause a flow of heat of 1 watt through that resistance, say, of a particular furnace wall or whole furnace. Hence, to determine the actual heat current or flow in watts through this resistance, we must divide the actual temperature drop for the furnace by this resistance (also in degrees), just as with electricity, to determine the current C, we divide the actual drop of voltage E by the resistance R, according to Ohm's law.

It is, therefore, quite correct and proper to express a thermal resistance in degrees and to use this quantity in subsequent calculations just as we use ohms in electrical calculations. The results will be quite correct numerically.

Quite similarly, thermal conductances can be correctly and properly measured and expressed in watts, and this number of watts can be used in calculations just as electrical conductances (in mhos) are used. The physical meaning of a thermal conductance in watts, say, of a complete furnace or any part of it, is that this is the number of watts which would flow when there is 1° difference of temperature between the two end surfaces. Hence, to get the actual flow in any given case, multiply this conductance in watts by the actual existing drop in temperature, just as, in accordance with Ohm's law, to get the flow of current C in any given case we multiply the conductance K by the drop of voltage E.

The thermal resistance and conductance of a furnace or any part of it are fixed quantities like its dimensions, and are independent of any flows of heat; just as the electrical resistance and conductance of a wire are fixed quantities independent of the current which may or may not flow. The currents of heat or electricity will be dependent upon the drop of temperature or voltage, and are determined by Ohm's law.

Of course, in so far as the temperature coefficients of the thermal resistivities or conductivities affect the values of the latter, the proper corrections must be made if they are sufficiently great to become important. But here the electrical analogy still holds good, because when a current flows it raises the temperature and this in turn changes the resistivity; whenever this is of importance (as in incandescent lamps, for instance, where this change amounts to about 50 per cent) it must be allowed for. But this is of the nature of a correction factor and should not be permitted to destroy or to cloud the very interesting and valuable result given us by this very close analogy, or the fundamental laws involved.

The advantage of using the thermal resistance and conductances in furnace calculations, instead of the heat losses through them, is that it simplifies the calculations, and in some cases even makes a direct calculation possible where troublesome trial calculations would otherwise have to be used. Furnace walls, made as usual of layers of different materials, are an illustration. The thermal resistance of each layer is calculated by itself; the total is then equal to their sum, and from this the heat losses can finally be calculated; the drop of potential in each layer is then determined from its resistance and the total flow, quite like the drop of voltage in various electric conductors connected in series. To make such a determination with conductances alone would be very tedious, if indeed possible at all.

It seems unnecessary to accompany the above with formulas and examples, as the calculations are in every respect quite analogous to the corresponding electrical ones, and are all extremely simple. One need only remember that thermal conductivities are expressed in watts for 1 in. (or centimeter) cube and 1° drop; that conductances are expressed in watts for 1° drop; that thermal resistivities are expressed in degrees drop for 1-in. cube and 1-watt flow; that resistances are expressed in degrees drop for a flow of 1 watt; and that the numerical values of the resistivities and corresponding conductivities are then the reciprocals of each other, as are also the values of the resistances and conductances.

The latter, being the important connecting link in this new system of units may be further illustrated by the following example. Inch units and centigrade degrees are implied.

Let the thermal conductivity of graphite be 3 watts. Assume a piece 6 in. long and 4 sq. in. section. The conductance of this piece will then be $3 \times 4 \div 6 = 2$ watts, which means that a heat current of 2 watts would flow for 1° drop between the ends. If this system of units is correct the reciprocal of this conductance must be the resistance in degrees, hence equal to 0.5° C. This means that the drop over the whole length would have to be 0.5° for a flow of 1 watt. In the above calculations it was found to be 2 watts for 1°; hence, it will be 0.5° for 1 watt; the resistance is, therefore, the reciprocal of the conductance. If the resistance is 0.5° the resistivity must be: resistance $0.5° \times$ section $4 \div$ length $6 = 0.333°$, which, it will be seen, is the reciprocal of the conductivity, 3 watts.

It would, of course, be more rational to follow electrical practice and create two new thermal units, one for resistance (and named the thermal ohm*) and the other for conductance (and named the thermal mho*) based on the two quotients of the two variables, watts and degrees, than to use those described above, which are based on making one of them unity so that the other one is numerically equal to their quotient. But until such a rational system has been devised and adopted the units here deduced will at least serve our purpose, even though it may at first require some mental effort to overcome one's natural dislike to measure a resistance in thermometer degrees and a conductance in power units.

There are still other resistance and conductance units which could be similarly deduced as, for instance, a length under unit conditions of flow, temperature and section; or a section under unit conditions of flow, temperature and length. They might even have some advantages over those recommended in this article. But the valuable analogy with Ohm's law would then appear to be destroyed, and this would require new laws to be formulated and learned.

It may be of interest to recall here the statement that Dr. Ohm based his famous electrical law on the corresponding and older one which had been found to be true for thermal flows.

Philadelphia, Pa.

Zinc Ore Dressing in Colorado--III.

By H. C. PARMELEE.

The Creede District.

Among the mining camps of Colorado none has had a more rapid and spectacular rise and fall than that of Creede in Mineral County. Practically unknown prior to 1890, the district had a railroad in 1891, experienced a boom in 1892, with an output of over $3,000,000 for the year, and collapsed in 1893 with the sudden fall in silver. Since that time the district has had varied fortunes and is now among those in which the greatest activity is confined to the dressing of base sulphide ore containing zinc and lead. In this district the mineral usually occurs in compact quartz veins, carrying galena and blende, the latter in the varieties known as rosin and black jack. Lead carbonate also occurs with the galena in some of the ores. Silver minerals and gold in small quantities are also common constituents.

The Amethyst Mill.

Of the three mills now in operation near Creede the Amethyst is probably the most interesting. The crude ore treated there averages about 5 per cent zinc and 10 per cent lead, with some silver and free gold. The silver minerals are closely associated with the silicious gangue, and there is a consequent heavy loss of silver in the tailing. The gold occurs free and may be seen on some of the tables treating the finer sizes of pulp. The lead and zinc minerals occur free enough to warrant coarse concentration with jigs, followed by intermediate treatment on

*These two very appropriate names were first suggested by Dr. A. E. Kennelly.

tables, finishing with slime. Quite an extensive plant is installed to save the latter. The average saving of the total value is nearly 85 per cent, the highest recovery being made on lead, next on zinc, gold and silver, the last suffering the greatest loss.

Ore is delivered at the head of the mill by an aerial gravity tramway, 1700 ft. long, connecting mine and mill. The line has 18 buckets carrying about 500 lb. of ore each, moving at about 200 ft. per minute, so that a bucket is dumped a little oftener than once a minute. This gives a capacity of about 125 tons in eight hours, which is the capacity of the mill for 24 hours. The tramming and crushing department is, therefore, operated only one shift.

At the tram loading station at the mine the ore is roughly sized by being passed over a grizzly with 1½-in. openings. These two classes are separately trammed and dumped into separate bins at the head of the mill, the coarse passing over a second grizzly before going to the crusher, and the fine going

FIG. 1.—THE AMETHYST MILL DISTRICT, CREEDE, COLO.

directly to the crushed ore bins. The bin capacity for crushed ore is 75-80 tons, or sufficient to run the mill two shifts.

Coarse ore is crushed through a jaw crusher reducing to 1½-in. size. From the crushed ore bin the ore is fed by automatic shovel feeder to the coarse rolls (14 in. x 27 in.), water being added as the ore leaves the feeder. The coarse rolls reduce the ore to about ½ in., and the pulp is then elevated by a 10-in. B. & B. elevator to the trommel screen line of four 3-ft. x 6-ft. trommels fitted with woven wire screen of 4, 5, 8 and 12 mesh. The first half of each trommel, except the last, is fitted with screen of larger wire than the second half, as shown in the table on top of next column.

The oversize of No. 1 trommel is returned to finishing rolls for regrinding, thence to the elevator and screen line. The undersize is passed along through the screen line, the oversize of the three succeeding screens being jigged, and the undersize of the last screen being classified for fine jig and table treatment, the coarsest class going to a jig and the others to tables.

Mesh and Wire of Trommel Screens.

Trommel No.	Mesh.	Wire Gage. 1st half.	2d half.
1	4	11	13
2	5	13	15
3	8	16	18
4	12	20	20

Jigging thus commences on —4 + 5-mesh material and continues to the coarsest size in the —12-mesh pulp. It is probable that jigging could be continued on smaller sizes, and less coarse work done on tables, as the minerals are quite free, especially the lead, and would make good jig products. There are four four-compartment jigs of the Harz type, with hutch discharge only. The first three are fed with screen oversizes as described before, and the fourth with the coarsest of the classified pulp. Lead concentrate is made in the first two compartments of all jigs, and middling in the third and fourth. Only jigs 1 and 2 (coarse) have tailing discharge which goes to waste, while jigs 3 and 4 (fine) are run without screens in the third and fourth compartments, so that the tailing water is almost clear and is sent to the slime plant.

Jig Data.

Jig. No.	Speed.	Mesh and Wire Gage in Compartments. 1.	2.	3.	4.
1	188	5-15	5-13	5-15	5-15
2	240	5-13	5-13	5-15	5-13
3	270	8-16	8-16	no	no
4	270	8-16	8-16	screen	screen

The middling from the third and fourth compartments of all jigs is reground in Chilian mills, of which there are two, 5 ft. in diameter, running at 30 r.p.m., fitted with 16-mesh woven wire screens. The roll rings are renewed every 80 days and the dies every 160 days, one die serving for two sets of rings. The Chilian discharge is elevated and combined with the undersize of the screen for classification.

The classifier used at the Amethyst mill is of the type known as the trough or shallow pocket classifier. The trough is rectangular in cross-section and has a pitch of about 1 in. per foot. Shallow pockets are placed at intervals in the trough, according to the number of classes which is designed to treat and the location of the machines on which they are to be treated. The pockets have the effect of momentarily retarding the flow of pulp in the trough, so that a classification can take place either with or without the use of hydraulic water. At the Amethyst mill no hydraulic water is used, although it is available for use in opening plugged spigots.

The pulps from the undersize of the screen line (12 mesh and finer) and from the Chilian mills (16 mesh and finer) are combined in the trough classifier just described. There are nine pockets in the trough, the discharges from which feed No. 4 (fine) jig and eight tables.

Card tables are used exclusively in this mill, there being 12 on classified pulp and three on slime. Lead concentrate is a finished product on all tables, being cut clean and free from the zinc middling, which is retreated on other tables where both lead and zinc concentrates are made. It will be observed that no zinc concentrate is made in this plant on the jigs or until the retreatment tables are reached. All concentrates are laundered directly to dewatering bins at the lower end of the mill, so that no handling is required except in loading for shipment.

The precautions taken at this plant for the recovery of mineral in the slime are worthy of special mention, as the slime recovery is high. The slime from Nos. 3 and 4 jigs and from 9 of the 12 tables treating classified pulp is laundered to three 8-ft. Callow thickening tanks, the underflow of which is treated on three Card slime tables. The overflow of the Callow tanks, the slime from all the tables which do not discharge into the tanks and the overflow from the dewatering bins are combined

and treated in the canvas plant. This is a recent addition to the mill and comprises six 50-ft. canvas tables 3 ft. wide, built on a gentle slope and covered with 10-oz. duck. The feed is distributed uniformly over all, but can be diverted from any one during the operation of cleaning, which is accomplished by a spray of pressure water directed by the operator. The accumulated slime is thus washed into settling boxes, of which there

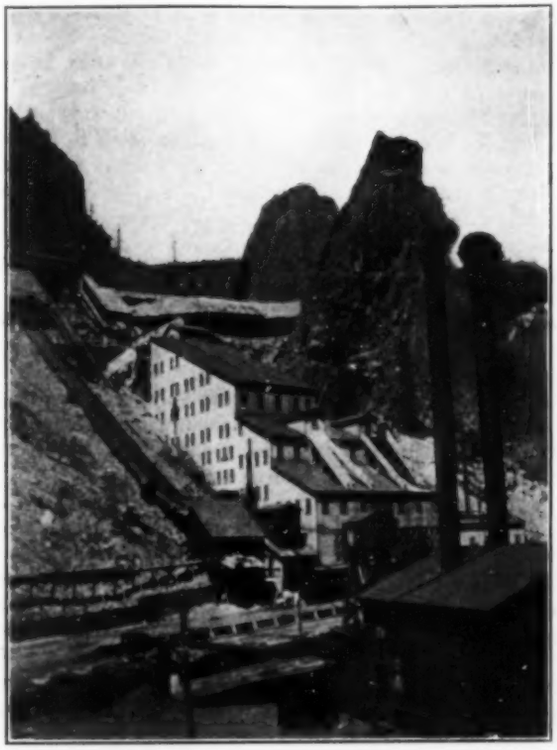


FIG. 2.—THE HUMPHREYS MILL DISTRICT, CREEDE, COLO.

are two sets arranged in compartments so that a compartment in one set can be cleaned out while the corresponding compartment in the other set is receiving the flow of slime from the tables. The overflow of these settling boxes is laundered to final deep settling tanks, which are cleaned but once in five months. One man is regularly employed on one shift to clean up the settling boxes by siphoning as much water as possible and then shoveling out the slime. It is dried in bins having steam pipes on the bottom and sides. As the slime is principally lead it is mixed with the lead concentrate for shipment.

The lead concentrate contains 70 per cent to 74 per cent lead; the zinc concentrate, 40 per cent to 45 per cent zinc, and the slime, about 35 per cent lead. The waste from the slime plant carries from 1 per cent to 3 per cent lead. No regular sampling of mine or mill products is done at this plant, the management satisfying itself with an occasional sampling of the entire mill.

Steam power with rope drive is used and has been found superior to the irregular electric power formerly available in the district. The power consumption is about 85 hp. From five to seven men are required per shift, in addition to the mining force and those engaged in hauling concentrates to the cars, a distance of about two miles.

The Humphreys Mill.

This mill is larger than the one just described, and if run at capacity would probably treat 250-300 tons per 24 hours; but being run at reduced capacity and for only one 10-hour shift per day, it is treating about 50 tons. The ore is practically the same as that treated in the Amethyst and the general process is the same. The equipment comprises crusher, coarse and fine rolls, automatic sampler, a double screen line of four trommels each, 12 four-compartment Harz jigs, three Chilian mills, trough hydraulic classifier and 12 Wilfley tables.

The jigs make lead concentrate in the first two compartments and zinc-lead middling in the last two. The latter are run with continuous hutch discharge, and with coarse screens and beds instead of without screens as at the Amethyst. The undersize of the screen line is not combined with the Chilian mill product, but each is treated on tables separately. Experiments have been made with various screens for the Chilian mills, trying different weaves of screen cloth and round punched sheet steel. The results indicate that while there is some preference as to weave of cloth, the round punched screen is superior to them, being more free in its action and having longer life.

The slime plant at this mill is not in commission. It contains reciprocating tables and several 80-ft. canvas tables similar to those at the Amethyst mill. The power consumed in milling is about 100 hp, which is generated by steam and transmitted by rope drive. The illustration shows the power house in the foreground, with rope drive extending across to the mill.

The Ridge Mill.

From the point of view of the nature of ore treated, and the high quality of zinc concentrate produced, this mill is one of the most interesting in the district. The same minerals occur in the Ridge ore as mentioned before, viz., galena, cerusite and rosin and black jack zinc. The ore is easily mined, and as practically all of the values are in a stiff clay the hard rock is sorted as waste and thrown over the dump. Large boulders of galena covered with zinc are frequently found in the clay.

The nature of the ore eliminates crushing, and the first step in the milling process consists in disintegrating the stiff clay in a revolving disintegrating screen of the American Concentrator Company type. As shown in the illustration (Fig. 3), which is a sectional view looking through the screen toward the discharge end, the machine consists practically of a revolving grizzly, jacketed with a screen and revolving in a trough of water. Heavy cast-iron hoods reduce the openings at each end, through one of which the ore is fed, and from the other the barren rock discharged. One-inch iron bars extend from end to end of the trommel and into the faces of the hoods. The screen jacket surrounds these bars and has 1½-in.

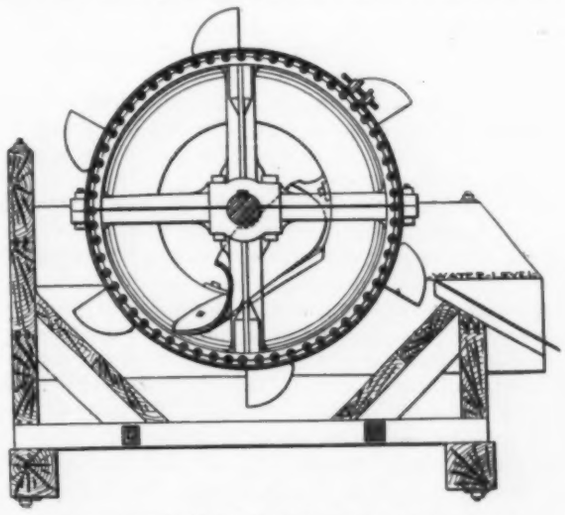


FIG. 3.—SECTIONAL VIEW OF DISINTEGRATING SCREEN.

round openings. On the outside of the screen jacket are bolted at intervals elevator cups, as shown in the illustration. The number of cups determines the quantity of feed and may be increased or decreased as needed. At the discharge end of the screen is a plow-shaped casting which, in its revolutions, picks up and discharges any pieces of rock which may have been fed with the clay. The rock is taken to the dump.

The ore, disintegrated by the revolution of the screen, falls through the openings into the trough, where it is picked up by the buckets and discharged onto an apron. It is then sized for jig and table work in the usual trommel screens to which it is elevated in the usual manner. The first trommel has 4-mesh screen cloth, the oversize of which is sent to rolls and again elevated. The undersize passes to a second trommel, which delivers two undersizes, one from each half of the trommel, and an oversize. The upper half of the trommel has 8-mesh screen, and the lower half, 6 mesh. The 8-mesh undersize is hydraulically classified, the underflow going to the fine jig and the overflow to the tables. The 6-mesh undersize goes to the medium jig and the oversize to the coarse jig.

There are three four-compartment Harz jigs having 5-mesh, 6-mesh and 8-mesh screen on the coarse, medium and fine jigs respectively. Hutch products only are made; lead concentrate in the first compartment, middling in the second, zinc concentrate in the third and middling in the fourth. Zinc concentrate is occasionally skimmed from the last compartment. The middling is returned to the elevator and rescreened without regrinding. The coarse, medium and fine jigs are run at 230 r.p.m., 240 r.p.m. and 260 r.p.m. respectively.

Three Wilfley tables of an early date are used and do good work. The first two make lead concentrate and middling, which latter is retreated on the third table where zinc and lead are separated. Most of the concentrating at this mill is done in the jigs, and the tables are not relied upon as much as in the other mills where no jig zinc concentrate is made.

Crude Ridge ore assays about 1.5 oz. silver, 0.01 oz. gold, 4 per cent to 6 per cent lead and 8 per cent to 12 per cent zinc. The proportion of the two base metals to each other is the reverse of that in the ore treated in the other mills.

The quality of the lead concentrate is indicated in the following assays of recent shipments:

Gold oz. per ton.	Silver oz. per ton.	Lead per cent.	Zinc per cent.
0.42	4.95	68.6	8.8
0.25	5.00	65.0	7.0
0.16	6.20	68.0	7.5
0.16	5.45	72.5	6.4

The zinc concentrate is of excellent quality, carrying from 50 per cent to 56 per cent zinc, 3 per cent lead, 0.05 oz. gold and 2.0 oz. silver.

The Ridge mill is provided with low-pressure water-power. The 20-in. pipe line has a drop of about 100 ft., and water is delivered through a 3-in. nozzle to a 4-ft. Pelton wheel at a pressure of 35 lb. per square inch. The mill has a capacity of 50 tons per 24 hours, but is run only one 10-hour shift and treats about 20 tons per day.

The three mills of the Creede district are now treating about 200 tons of ore per day in plants having a capacity of nearly 450 tons. The ratio of concentration is about 8 or 10 into one. It is probable that the dressing of base ore will continue to be an important feature here, and with steady development underground will probably increase in magnitude. One of the needs of the district is cheap and reliable electric power, such as is supplied to other districts in Colorado. At present coal is expensive, especially when hauled from the cars to distant plants, the latter haul often adding 50 per cent to 100 per cent to the original cost of the coal.

Settling dust from furnace gases is accomplished principally by reducing the velocity of the gases and allowing sufficient time for the dust to settle. Reduction of the temperature is not a necessary element, in fact, a higher temperature might be of benefit as the specific gravity of the gases would be reduced more by the temperature than would that of the dust particles. It has been demonstrated that if the velocity of gases is reduced to about 3 ft. per second there will be practically complete settling in less than 500 ft. in about 8 minutes.

Corrosion of Metallic Food Containers.*

By Edward Gudeman, Ph.D.

With a product for analysis one will be satisfied to call it a c. p. product, if the amount of total impurities is 0.001 per cent or less. Atomic weights generally do not go beyond the second decimal place, and most chemists in figuring analytical results use only the first decimal figure. In food chemistry, however, a quantity of 0.001 per cent is often considered a tremendous quantity, especially if it is a metallic impurity or an antiseptic or coloring ingredient, and I cite some cases to give an idea how far some food chemists and food officials carry this "foolish accuracy," not to use a stronger term.

Within the last few weeks there was submitted to me from outside of this State (Illinois) some carbonated water, condemned as unwholesome and poisonous, due to its lead contents, and the distributers were given five days to take same out of the market. Analysis showed the amount of lead to be from 0.00003 per cent to 0.00005 per cent. Another similar product was found to be free of lead as it was put into the container, but contained about 0.0003 per cent lead after three days. The lead in this case came from the metallic button used to seal this special kind of bottle. In another case the trace of lead was found to be due to solvent action on the pipes used in the apparatus and the manufacturer was given 10 days to take out the whole machinery and replace it with such machinery as would give a lead-free product.

The stronger the corrosion of the metal, the quicker the solvent action. In the case where the metallic stopper was the cause of the lead entering the beverage it was found that the button used to hold the rubber cap averaged in new buttons about 18 per cent of lead. The old, corroded button was found to contain about 8 per cent of lead. The total amount of lead and tin that actually went into solution could not be exactly determined, as such buttons vary in weight, but it was strongly indicated that approximately twice as much lead went into solution as of the tin. It was also found that the solvent action of carbonated waters on lead alloys was about the same as on the lead itself.

As another case showing to what foolish extremes of accuracy some food chemists will go I cite the following: In one of our neighboring States the distributers of a food product were given 10 days' notice to clear the market of a food product said to contain a quantity of sulphur dioxide or its compounds. On request I investigated this matter, and the chemist who made the analysis stated as follows: 200 grams of the product were taken for analysis, using the official methods, the distillation was continued until the distillate amounted to 2500 cc. To obtain complete oxidation, with bromine, this was concentrated down to 50 cc, placed into a test tube and the sulphates precipitated with barium chloride. As no visible precipitate appeared, the solution was allowed to stand about 18 hours, the clear liquid carefully decanted and the bottom of the test tube examined with a magnifying glass, showing a thin film. The film was considered as barium sulphate due to the sulphurous oxide in the original product, and the product was condemned and the distributers and manufacturers given a short time to dispose of the stocks on hand.

Duplicating the work quantitively it was found that the film would approximate 0.000001 per cent of the amount taken for analysis. Under the official standard for sulphurous oxide, pertaining to imported beverages, a maximum of 50 milligrams free sulphurous oxide per kilogram is permissible.

Parties have been cited into the courts to defend the presence of a faint trace of nitrogen oxide in starch. The testimony showed that the amount present was less than found in the atmosphere under normal conditions and the defendants were discharged.

*A paper read at the Chicago meeting of the American Electrochemical Society. An abstract of this paper was already given on page 635 of our November issue. But in view of the interest that attaches to the subject the paper is herewith printed in full.—Editor.

Within the last month Federal Inspection Decision No. 126 places the maximum amount of tin allowed in food products at 300 milligrams per kilogram being 0.03 per cent. Tin has been one of the substances most extensively used to protect iron from corrosion and until the issuing of the last Federal Inspection Decision was not legislated against; in fact. standards were accepted for the amount of tin required on tin plate to make it acceptable to be used for the manufacture of food containers.

Tin, copper, lead, silver and iron are very soluble metals (especially when traces are taken into consideration) in the liquids used with canned food products and in the peculiar liquids produced in the decomposition of foods. It is immaterial how the food itself is protected; it will undergo some decomposition, not necessarily putrefaction nor producing an unwholesome product. This decomposition is often a very slow one, not noticeable until months or years have passed since the product was put into the hermetically sealed container.

Among the two most commonly found products of decomposition are sulphuretted hydrogen and carbonic acid. Carbonic acid is one of the strongest corroding acids known to me, especially when acting under pressure on metals in the presence of water. Some 18 years ago I was interested in the process of directly corroding metal lead with carbonic acid. Treating finely divided lead with carbonic acid under pressure would change the lead into carbonate in a very short time, the process being now in use for the manufacture of carbonate of lead, white lead, direct from the metal. The finely divided lead or lead sheets are subjected to water or steam under a pressure of carbonic acid. Carbonated beverages are always under carbonic acid pressure varying from 5 lb. to 125 lb. per square inch, and the decomposition of food products giving rise to carbonic acid produces a pressure, and thereby solvent action is increased. This no doubt explains why food products placed in vacuum containers are less liable to deterioraton.

Tin is a good protecting medium for iron under two conditions: The coating must be perfect and the contact between the tin and the iron must be perfect. I have never seen a piece of tin plate where both of these conditions were met with, and very few where either of these two conditions was fulfilled.

There are four main reactions that take place with food products, due to the character of the foods themselves or to their decomposition under normal conditions of ageing. There is either oxidation or reduction, or both, and the action of acids or alkalis, or both. A perfect container will be one that will withstand all of these reactions.

One cannot tell with certainty which reactions will take place nor how they will take place nor when. Iron is porous and in tinplating the pores are not filled or coated. Tin also is porous, and this porosity must be taken into consideration. A perfect contact between the tin and the iron is of the greatest importance, using sufficient tin to coat all the superficial area of the iron. With such a tin plate, the corroding action on the tin need only be considered. After a large series of trials this object was attained, but the process is not commercially successful, as the cost of the container is a vital consideration, being about twice the value of the present tin can with its contents.

Iron and steel can be impregnated so as to have the resisting properties of gold. Steel springs, knives, surgical instruments, etc., look like gold and have the insolubility of gold, with all the characteristic properties of tempered steel. The process is simple. I made an amalgam of gold and impregnated the iron article with such liquid amalgam, then by heat vaporized the mercury and obtained an iron-gold article that withstood the corroding action of acids, exactly like gold. The amount of gold required was very small, about 2 oz. of gold to a ton of iron being sufficient, figuring this from the small experimental samples made.

The serious objection to this process was that the amalgam of gold had to be liquid and it was necessary to vaporize about 99½ lb. mercury for every ½ lb. of gold used, an amalgam of over ½ per cent not working satisfactorily. Experimentally the process is a success, but commercially the cost of operation is such that the cost of such a plate will be from 12 to 15 times the cost of a tin plate, making the cost of the gold-iron container exceed, often, the cost of the present container with contents.

Other liquid alloys were tried and some very peculiar results obtained. Copper impregnated with sodium amalgam was found to resist the action of acids and water to a greater extent than the metals themselves. I have no explanation to offer why a copper-sodium plate should resist acid more than the copper itself. The resistance of such a plate to water being greater than the resistance of sodium itself can be explained on the theory of the dilution of the sodium with the copper, thereby protecting the sodium from coming into contact with the water.

Very peculiar results were obtained when impregnating copper with antimony amalgam. It was found that such antimony-impregnated copper greatly resisted the action of dilute nitric acid, the amount of copper that went into solution from the surface contact being not over 10 per cent of the amount that went into solution from the edges. The acid penetrated from the edges toward the center, and the rapidity of the solvent action of the acid from the edges toward the center was visibly noticeable. Strength of acid, temperature and time are factors that materially influence this action. Protecting the edges by dipping into paraffin showed conclusively that the corroding action of nitric acid on copper was greater than on the same amount of copper impregnated with antimony.

Very extensively used for food containers are the so-called enameled cans. They are not real enameled cans, but simply metal plates on which an enamel has been baked. Real enamel should become part of the whole and not simply a superficial protecting coating that can be separated by mechanical means. Experiments were made to impregnate enamels in iron plates, but the high pressure required makes the process too expensive to be commercially developed now. The enamel was forced into the iron plate at a pressure of about 50,000 lb. per square inch. The superficial excess coating of enamel was scraped off the plate and the impregnated plate was subjected to a heat sufficient to melt the enamel. Enamel-impregnated iron plates withstood the action of acids and alkali fully as well as glass.

There were many difficulties found in using enamels and experiments were then made to impregnate iron plates with glass. The results were most satisfactory, and until the process and products are fully protected no details can be given.

The best results were so far obtained when using a very fusible lead-glass. Such impregnated glass-iron plates were found to resist the corroding action of food juices fully as well as the glass itself. The amount of lead that will go into solution has not yet been determined. Whether such a container will meet the requirements of food officials and food chemists, who condemn food products containing 0.00003 per cent of lead and who consider about 0.000001 per cent of sulphur dioxide as unwholesome, remains to be seen.

CHICAGO, ILL.

Fine Grinding.—Apropos the discussion recently engaging so much attention from South African metallurgists in regard to the efficiency of the stamp-tube mill combination, it is interesting to note a recent statement in the *South African Mining Journal*. It has been suggested before that classification and regrinding in stamps would effect the same result better than the stamp-tube mill combination, but the suggestion has not been received favorably. Now, however, according to the authority mentioned, "the New Kleinfontein mill is being prepared for crushing operations, in which the work of the tube mill, now almost an essential of the metallurgical scheme of the Rand, will be done much more efficiently, cheaply and conveniently by stamps alone."

The Equilibrium of the System Consisting of Calcium Carbide, Calcium Cyanamide, Carbon and Nitrogen.

By M. de Kay Thompson and Robert H. Lombard.

(*Concluded from page 620.*)

The experiments described in the first part of this article, published in the November issue, show the method of working, and for the sake of brevity the detailed record of experiments 9 to 17 will be omitted. The final results are given in table 3 on top of the next column on this page.

EXPERIMENT 18.

The disadvantage of the Arsem vacuum furnace for experiments of this kind is that the volume is so great that it takes quite a long time for the solid substance to absorb or evolve enough gas to reach equilibrium pressure. It was therefore decided to try an experiment in the Heraeus furnace that had been used as an auxiliary furnace for calibrating the Wanner pyrometer. Of course, this could be used only at a comparatively low temperature. It was glazed only on the inside, and it was anticipated that it would leak.

The charge was contained in a graphite boat 1.5 in. long. It consisted of 6.14 grams of a mixture of calcium carbide and calcium chloride containing 10 per cent of the latter. The ends of the porcelain tube of the furnace were closed with rubber stoppers; the stoppers and ends of the tube were shellaced. A small glazed porcelain tube for holding the thermo-electric couple extended through the larger tube and the two stoppers. The junction was pushed in until it was opposite the graphite boat in the middle of the furnace.

Aug. 10 at 5 p. m. the furnace was pumped out to a pressure of 1 mm and left till Aug. 11, 16 hours later, when it had leaped to 13.1 mm. The furnace was then heated to about 1070° and pumped out a number of times, after which the gases remaining were swept out with nitrogen. While still hot nitrogen was admitted to 110 mm. Absorption began to take place rapidly and the run was begun.

Time.	Pressure.	Temperature.
2:08 p. m.	110	....
2:10 " "	32	1074
2:13 " "	23.2	1074
2:20 " "	21.5	1074
2:25 " "	21.6	1074
2:32 " "	22.1	1074
2:40 " "	22.8	1077
2:45 " "	23.2	1077

It seems that here the absorption of nitrogen at first overbalanced the leakage. The following day minima of 26.8 mm, 30 mm and 31.5 mm were obtained, starting with a pressure of 145,165, and between 100 mm and 200 mm respectively. The heat value, of course, would be obtained by starting very little above the equilibrium pressure, so that there would not be so much time for leakage to take place.

Although these determinations were somewhat in error from the leakage, this error was evidently not great, as the pressures agreed well with those obtained in the vacuum furnace. The temperature measurements were more reliable than the others, as they were made directly with the thermo-electric junction. The lowest pressure, 21.5 mm, is presumably the most reliable. The result of this run is therefore 21.5 mm at 1074° C.

The results of the above experiments are collected in Table 3. and are plotted in Fig. 5. The ordinates are pressures in centimeters of mercury and the abscissæ are the temperatures in centigrade degrees. The pressures are not reduced to zero degrees, as this correction is much less than the experimental error.

It is remarkable that the relation between the temperature and pressure is linear through such a great range in temperature. It would seem that at a lower temperature the line representing this relation must become concave upward, and it is hoped that in the near future it will be possible to carry out some experiments on this equilibrium below 1000° to test this point.

TABLE 3.

No. of experiment.	Pressure in cm. of Hg.	Temperature centigrade.
4	27.0	1263
6	22.4	1223
7	30.8	1297
8	37.8	1378
10	12.5	1160
10	16.1	1196
10	27.3	1278
11	28.3	1308
12	2.3	1053
12	6.3	1117
13	6.8	1114
15	44.4	1448
16	38.8	1372
17	37.5	1385
18	2.2	1074

In order to compute the heat Q, evolved by the reaction

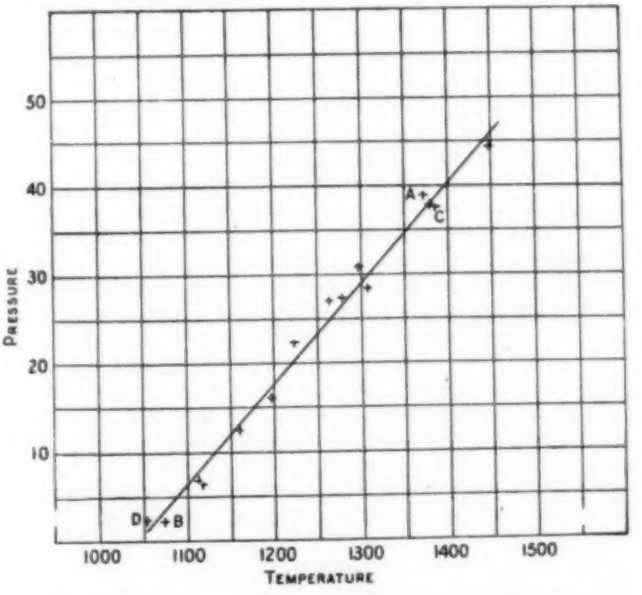


FIG. 5.—CHANGE OF HEAT OF REACTION WITH TEMPERATURE.

$CaC_2 + N_2 \rightleftarrows CaCN_2 + C$, taken from left to right, it is not allowable to use the integrated van't Hoff equation

$$4.57 \log \frac{p_2}{p_1} = Q\left(\frac{1}{T_1} - \frac{1}{T_2}\right),$$

because this equation is derived on the assumption that Q is constant, which is not the case in this reaction. If Q is computed from the differential equation

$$\frac{dp}{dt} = \frac{Q}{T\,(v - v_1)}$$

in which T is the absolute temperature, v is the volume of the carbide and nitrogen and v_1 is that of calcium cyanamide and carbon on the other side of the equation, and $\frac{dp}{dt}$ is the pressure gradient at the temperature T, Q is found to have the values given in Table 4. The volume v_1 is negligible in comparison with v. Omitting v_1 the equation may be put in the following form:

$$Q = \frac{RT^2}{p}\frac{dp}{dt}$$

The value of R is 1.99 calories and the unit in which p is expressed cancels out.

TABLE 4.

Showing the Change in the Heat of the Reaction with the Temperature.

Temperature Centigrade.	Q in Gram Calories.
1100	63,900
1200	27,000
1300	18,800
1400	15,200

The value of $\frac{dp}{dt}$ is, of course, constant for every temperature

The line as drawn in Fig. 5 seems to represent the observations better than any other. The maximum variation in the slope would be represented by the lines *AB* and *CD*. From the value of $\frac{dp}{dt}$ obtained from these lines, the heat was calculated for 1200° and found to be 29,800 and 25,800 calories, respectively. This shows that the value calculated from the best representative line, 27,500, cannot be in error by more than 2000 calories, or 7 per cent.

It seems evident from the fact that the heat effect is greater the lower the temperature that it would be advisable to work at as low a temperature as possible in the fixation of nitrogen by this method, because less heat would be required from external sources. This would be true only if the velocity of the reaction did not decrease with decreasing temperature, for this might more than balance the benefit due to the heat of the reaction.

In view of the enormous increase in the heat of the reaction with decreasing temperature, as computed from these measurements, it is desirable that the heat should be measured in some other way. This might be done, as suggested by Mr. S. Peacock, by combustion experiments in a bomb. This would give indirectly the heat of the reaction at room temperature, which would be expected to be much larger than the values calculated above.

The free energy increase of the reaction

$$CaC_2 + N_2 \rightleftarrows CaCN_2 + C$$

at 1450° C. taken from left to right is

$$\Delta F = -RT \log \frac{76}{46} = -1714 \text{ gram calories.}$$

At 1100 C., $\Delta F = -6700$ calories.

The errors in the temperature measurements are probably the main cause of the deviations of the single determinations from the best representative line. If all the error is assumed due to the temperature measurements the average deviation of a single point from the line is only 12°, and the largest deviation is 19°. These are not greater than the experimental errors in the temperature measurements.

The thermo-electric couple had been cut off several times since the original calibration. It was subsequently calibrated as follows:

TABLE 5.

Date.	Temperature.	Millivolts at date given.	Millivolts at original calibration.
July 2.	Sulphur point.	3.60	3.60
Aug. 13.	Copper point.	10.08	10.19

The original calibration had therefore remained practically constant throughout the experiments.

After each experiment for determining the equilibrium pressure, the furnace walls were covered inside with a white powder. A similar result was obtained in determining the equilibrium of the reaction $CaO + 3C \rightleftarrows CaC_2 + CO$. It was noticeable, however, that there was more powder formed with calcium cyanamide for a given temperature and given duration of heating than with calcium carbide. It therefore seemed probable that calcium cyanamide is volatile, especially after it was found that the white powder gave a yellow precipitate of silver cyanamide on adding silver nitrate to an ammoniacal solution of the powder. It was therefore thought desirable to make an analysis of this material.

Some of the pure white powder was analyzed for nitrogen by the Kjeldahl method, taking about one gram for analysis. The digestion with chemically pure sulphuric acid was continued for five hours near the boiling point; immediately after, potassium permanganate was added. The two determinations gave 12.38 per cent and 11.67 per cent nitrogen, corresponding to 34.3 per cent calcium cyanamide.

To get a check on this value the calcium cyanamide was determined by the method of Perotti.[5] This method consists in allowing the substance to stand over night in water, filtering and washing, adding ammonia, and then an excess of silver nitrate. A yellow precipitate of silver cyanamide is produced which is filtered off and the excess of silver nitrate is titrated with ammonia sulphocyanate. The sample taken was not perfectly white, but from its gray color evidently contained some carbon which had also distilled to the wall of the furnace.

The residue obtained on dissolving the white powder in water was of a dirty gray color. It was evidently carbon and calcium oxide, for (1) it dissolved slightly in a fresh portion of water and gave a distinct precipitate when ammonia and ammonium oxalate were added; (2) on treating some that had been exposed to the air with hydrochloric acid it dissolved partly with effervescence, leaving a black residue behind; and (3) when the hydrochloric acid solution was neutralized with ammonia and ammonium oxalate was added an abundant precipitate was formed. The analysis of the water solution for calcium cyanamide gave 38.59 per cent, 39.52 per cent and 38.84 per cent, corresponding to 13.51 per cent, 13.84 per cent and 13.60 per cent nitrogen.

An analysis of a sample of the same material used in the Kjeldahl determinations gave only 25.96 per cent calcium cyanamide, corresponding to 9.09 per cent nitrogen. Since this did not agree with the determinations by the Kjeldahl method, or with the values obtained by the same method for other samples, analyses were made by the Perotti method of two more samples. The perfectly white material used in the Kjeldahl method could not be checked again as it had all been used up. The results of the analyses of the powder obtained in Experiment 16 were 15.96 per cent and 15.35 per cent calcium cyanamide, corresponding to 5.59 per cent and 5.38 per cent nitrogen. Analyses of the material obtained in Experiment 17 gave 29.10 per cent and 28.95 per cent calcium cyanamide, corresponding to 10.19 per cent and 10.13 per cent nitrogen. Experiments 16 and 17 were made at about the same temperature and still the amount of calcium cyanamide in the powder distilled to the cold part of the furnace was quite different.

Two determinations of calcium in the white powder used in the Kjeldahl determinations were made by first destroying the organic matter with fuming nitric acid and then decomposing the calcium nitrate by ignition in a platinum crucible and weighing as the oxide. The fuming nitric acid had to be added very slowly, as it attacked the white powder with great violence, sometimes causing it to flame. The results were 39.93 per cent and 39.27 per cent calcium, the average of which is 39.60 per cent. From this and the fact that the content of nitrogen was 12.1 per cent the amounts of calcium oxide and of calcium cyanamide in the first sample of white powder analyzed were as follows:

Calcium cyanamide	34.5	per cent.
Calcium oxide	30.5	" "
Unaccounted for	35.0	" "

Time did not permit of making a combustion analysis to see how the carbon checked up. This sample apparently contained no free carbon, for it was of a cream-white color.

Since calcium cyanamide was present in the powder distilled

[4] *Trans. Am. Electroch. Soc.* 15, 197 (1909).

[5] *Chemisches Centralblatt* 2, 1058 (1905).

to the walls of the furnace, when heated with carbon it should produce calcium carbide with the evolution of nitrogen. In order to see whether this actually takes place 0.4 gram of the powder was mixed with 0.9 gram Acheson graphite and heated in the vacuum furnace. The results are the following:

EXPERIMENT 14.

		Kilowatts.	Pressure.
July 11.	11:00 a. m.	...	2.8
	11:12 " "	0.33	10.0
	11:20 " "	0.78	...
	11:35 " "	...	19.0
	11:35 " "	1.27	...
	11:45 " "	...	23.8
	11:45 " "	1.88	...
	12:00 m	...	28.8
	12:00 m	2.7	...
	12:20 p. m.	...	67.0
	12:20 " "	5.3	...
	12:35 " "	8.0	...
	12:45 " "	...	183
	2:15 " "	...	195

The temperature was measured at 12:35 p. m.; with the current in the Wanner 6.1 units the temperature measured 1287° C.

On opening the furnace after this run no white deposit was found on the walls, and the material in the crucible seemed to be hardly changed in appearance. It gave no test for calcium carbide with an ammoniacal cuprous chloride solution, nor for cyanamide with silver nitrate. It is to be noticed, however, that a great deal more gas was evolved from this mixture on heating than on heating a sample of carbide, from which occluded gases always escape. For example, in Experiment 15, on heating carbide to 1450° C., these gases produced a pressure of 127 mm, which is considerably less than the 195 mm produced from the white powder and carbon at 1287°.

It therefore seems certain that the large amount of gas evolved in Experiment 14 was partially due to a chemical reaction and not to occluded gases alone. A possible explanation of the absence of carbide in the crucible is that some oxygen got into the furnace. This would be changed to carbon monoxide, which would decompose any carbide formed. The result of the experiment is, however, not conclusive.

Since calcium oxide was found in the white powder, an experiment was carried out to see to what extent lime would distil when heated alone in a vacuum. A graphite crucible was filled with pieces of Mercks lime about the size of a pea, and was heated in a vacuum for three hours at 1531°. On opening the furnace it was found covered on the inside with a thin grayish white powder. On collecting this it filled about ½ in. of an ordinary test tube, and was placed in a desiccator several days before testing. It was then dissolved in hydrochloric acid and tested for calcium with ammonia and ammonium oxalate; a large amount of precipitate of calcium oxalate was formed. This experiment, therefore, shows that lime distils at about 1500° in a vacuum. This doubtless is the cause of the thin layers always found in the experiments on the equilibrium of the reaction $CaO + 3C \rightleftarrows CaC + CO$. It was not known at the time of the determination of this equilibrium that lime is distilled in the neighborhood of 1500°.[4]

3. Summary.

The equilibrium pressure of nitrogen in the reaction

$$CaC_2 + N_2 = CaCN_2 + C$$

was determined from both sides for a number of temperatures between 1050° C. and 1450° C. The results are plotted in Fig. 5. It was found that the heat of the reaction varies enormously with the temperature. The values calculated from the plot in Fig. 5 are given in Table 4.

The free energy increase of the reaction taken from left to right is −1714 gram calories at 1450° C. and −6700 calories at 1100° C.

Calcium cyanamide distills to the cold parts of the furnace at temperatures as low as 1050° C. Pure lime distills appreciably in the neighborhood of 1500° C.

Electrochemical Laboratory, Rogers Laboratory of Physics,
Massachusetts Institute of Technology,
Boston, Mass.

[4] This Journal, 8, 3, 27 (1910).

A Method for Increasing the Calorific Power of Washington Lignites.*

By H. K. Benson.

Many of the coal deposits of Western Washington are found in localities where the rocks have undergone only a moderate amount of folding and without any shearing movement. This is especially true of the Puget Sound basin proper and in these places the coal is lignitic in character.

The most important lignite fields are the Renton-Cedar Mountain and the New Castle-Issaquah fields in King County and these are the only ones that are being operated on a large scale at the present time. Extensive lignite deposits also occur in Thurston, western Lewis and Cowlitz Counties, but are not mined to any extent on account of the low heating value of the coal.

In order to ascertain the effect of drying and partial distillation on the heating value of some of the low grade lignites of Washington, the following experiments were undertaken in the laboratory of industrial chemistry, at the University of Washington, under the direction of the writer.

Four samples of run of mine coal were obtained for the experiments—one from Tenino in Thurston County; one from the Seattle Electric Company's mine at Renton in King County, taken from veins Nos. 2 and 3; one from the Denny-Renton Clay & Coal Company's mine at Renton, representing vein No. 1, and one from Centralia in Lewis County, obtained from the Chlopeck Fish Company, of Seattle, who use it in their gas producer. The above coals are what are commonly known as black lignites in Washington. The United States Geological Survey classifies them as sub-bituminous coals.

The coal was crushed to pea size and carefully sampled. Lots varying from 16 to 30 pounds were weighed out and placed in a large, horizontal, cylindrical iron retort, which was then sealed and heated with Bunsen burners placed underneath it until the bottom became a low red heat. The heat was applied until the gas ceased to come off in appreciable quantities. This took from 12 to 18 hours on account of the poor facilities for heating the retort.

The gas was passed through two large bottles, then through a condenser, then through water, and finally through a meter. It was sampled and analyzed. Most of the by-products condensed in the first two bottles. They consisted mainly of the water present in the coal, with some tar and a little ammonia.

Results of Experiment with Tenio Lignite.

Weight of coal placed in retort	= 30 pounds.
Weight or resulting produce	= 21 pounds 5¼ ounces.
Weight of by-products	= 6 pounds 5½ ounces.
Volume of gas	= 28.3 cubic feet.

Proximate Analysis of Tenino Lignite Before and After Drying.

	Before drying.	After drying.
Moisture	13.4 %	1.52%
Volatile matter	31.7 %	18.08%
Fixed carbon	34.17%	54.3 %
Ash	20.73%	26.1 %
Sulphur	1.34%	1.84%
Heating value per pound of coal	8525 b.t.u.	10,320 b.t.u.

The by-products contained 3.03 grams of ammonia and 147 grams of tar separated out. The tar had a specific gravity of .910.

*Abstract of paper submitted for the Kerl prize in Industrial Chemistry by Charles R. Fettke.

Analysis of Gas Given Off by Tenino Lignite.

Carbon dioxide	31.6%
Illuminants	2.3%
Oxygen	2.6%
Carbon monoxide	6.0%
Methane	37.1%
Hydrogen	7.6%
Nitrogen	12.8%
Heating value per cubic foot	488 b.t.u.

There was a decrease in weight of 28.8% in drying the coal and its heating value was increased 21.1%.

Results of Experiment with Seattle Electric Company's Lignite, Representing Veins 2 and 3, at Renton.

Weight of coal placed in retort = 16 pounds.
Weight of resulting product = 10 pounds 12¼ ounces.
Weight of by-product = 3 pounds 9¼ ounces.
Volume of gas = 21 cubic feet.

Proximate Analysis Before and After Drying.

	Before drying.	After drying.
Moisture	12.2%	1.4 %
Volatile matter	34.0%	25.53%
Fixed carbon	42.9%	56.97%
Ash	10.9%	16.1 %
Sulphur	.73%	.94%
Heating value per pound of coal	11,798 b.t.u.	11,735 b.t.u.

The by-product contained 3.49 grams of ammonia and 372.5 grams of tar separated out. The tar had a specific gravity of 9455.

Analysis of Gas.

Carbon dioxide	19.4%
Illuminants	2.4%
Oxygen	2.0%
Carbon monoxide	6.4%
Methane	26.4%
Hydrogen	23.9%
Nitrogen	18.5%
Heating value per cubic foot	433 b.t.u.

There was a decrease in weight of 33% in drying the coal and no increase in heating value. The heating value remained practically the same.

Results of Experiment with Denny-Renton Clay & Coal Company's Lignite, Representing Vein No. 1, at Renton.

Weight of coal placed in retort = 20 pounds.
Weight of resulting product = 13 pounds 15¼ ounces.
Weight of by-products = 4 pounds 6½ ounces.
Volume of gas = 21.3 cubic feet.

Proximate Analysis Before and After Drying.

	Before drying.	After drying.
Moisture	11.3 %	1.62%
Volatile matter	27.7 %	15.77%
Fixed carbon	37.6 %	51.55%
Ash	23.4 %	31.06%
Sulphur	.59%	
Heating value per pound of coal	8930 b.t.u.	9819 b.t.u.

The by-product contained 6.05 grams of ammonia and 275 grams of tar separated out. The tar had a specific gravity of .935.

Analysis of Gas.

Carbon dioxide	20.0%
Illuminants	2.1%
Oxygen	4.9%
Carbon monoxide	6.8%
Methane	24.7%
Hydrogen	22.9%
Nitrogen	18.6%
Heating value per cubic foot	407 b.t.u.

There was a decrease in weight of 30.20% in drying the coal and an increase of 10% in the heating value.

Results of Experiment with Centralia Lignite.

Weight of coal placed in retort = 20 pounds.
Weight of resulting product = 12 pounds 3¾ ounces.
Weight of by-products = 5 pounds 10¾ ounces.
Volume of gas = 24.5 cubic feet.

Proximate Analysis Before and After Drying.

	Before Drying.	After Drying.
Moisture	18.4%	1.4%
Volatile matter	29.3%	16.71%
Fixed carbon	36.7%	56.87%
Ash	15.6%	25.02%
Sulphur	1.76%	
Heating value per pound of coal	9086 b.t.u.	10,240 b.t.u.

The by-products contained 3.68 grams of ammonia and 312 grams of tar separated out. The tar has a specific gravity of 0.940.

Analysis of Gas.

Carbon dioxide	26.6%
Illuminants	1.6%
Oxygen	3.2%
Carbon monoxide	5.6%
Methane	29.2%
Hydrogen	19.1%
Nitrogen	14.2%
Heating value per cubic foot	428 b.t.u.

There was a decrease in weight of 28.8 per cent in drying the coal and an increase of 12.7 per cent in the heating value.

Results Calculated for 100 lb. of Coal.

	Tenino Lignite.	Seattle Electric's Lignite.	Denny-Renton Clay & Coal Co.'s Lignite.	Centralia Lignite.
Weight of coal taken in pounds	100	100	100	100
Weight of dried product in pounds	71.13	67.29	69.75	61.15
Increase in calorific power	21.1%	0.00%	10.0%	12.70%
Weight of ammonia in pounds	0.0223	0.0481	0.0667	0.0406
Weight of tar in pounds	1.08	5.13	3.03	3.44
Volume of gas in cubic feet	94.2	131.3	106.5	122.5

The foregoing results show a considerable increase in calorific power of low-grade lignites through partial destructive distillation with the exception of the Seattle Electric Company's lignite, in which case there was no change in heating value. However, this coal already had a high heating value for a lignite before it was dried. Partial distillation in this case resulted in increasing the ratio of volatile matter from 5 to 9, while in the other cases this ratio was increased from 5 to 16. The best results were obtained with the lignites from Tenino and Centralia, which run very high in moisture and are low in heating value.

Of the by-products the tar is probably the only one that would be of any economic value. From 3 per cent to 5 per cent of the weight of the coal taken was received as tar. The ammonia is produced in such small quantities that it would probably not pay to save it. From 1 cu. ft. to 1¾ cu. ft. of gas is also obtained per pound of coal used. This gas would be worth nearly as much as ordinary coal-gas and this would make the gas recovered from one ton of coal worth about $2. It could be used as fuel to dry more coal, or it could be used in a gas engine, or it could be sold.

Summary.

1. Analyses of lignites show high moisture and the volatile matter nearly equal to fixed carbon.

2. After partial distillation the moisture content is greatly reduced and the relative amount of fixed carbon increased.

3. A heating value of lignites is increased from 0.00 per cent to 21.1 per cent.

4. By-products recovered per ton are as follows:

Coal gas (b.t.u. = 430-488 per cu. ft.) . . 2000-2600 cu. ft.
Ammonia (NH_3) . . . . . 0.8-1.2 lb.
Coal tar . . . . . 100-20 lb.

5. The extended range of utility for these lignites and the commercial value of the by-product suggest this method as an improvement over the means heretofore used for utilizing low-grade lignites.

Laboratory of Industrial Chemistry
University of Washington,
Seattle, Wash.

The Smelting of Lead-Antimony Ores.

By A. W. Hahn.

As far as my knowledge goes the small smeltery located in the State of Queretáro, Mexico, and two days ride east of the city of the same name, is the only one smelting lead-antimony ores and making a saving at one operation of both the lead and antimony.

The property is owned by O. y T. Braniff, of Mexico City, and the mines are among those first worked by the early Spaniards in Mexico. The surface ores were clean lead carbonates, rich in both lead and silver, and easily treated. At depth the character of the ores changed, sulphides appeared and the lead was replaced to a greater or less extent by antimony. It is these ores which are being treated to-day, and in 1907, when I was connected with the company, at a profit.

The smeltery is located on a hillside, at the mouth of the main working tunnel of the mine called Socavón. The ore is trammed from the hoist station, by contract, to the ore-picking level, where it is dumped over a 1-in. grizzly. Here the ore is carefully picked and sorted into "gabarro," 4-in. size; "granzon," 1-in. to 4-in. size; "arrenillas," fines and waste. The 4 in. and 1 in to 4 in. go to separate bins; the fines go to the briquetting yard, and waste to the dump. Sulphides are separated into coarse and fines; the coarse going to the roast heaps direct, and the fines are first briquetted and then roasted. Ore is also received from a mine two hours' packing distance from the smelter; this ore has already been sorted, and is of two classes, raw carbonates and roasted galena, both classes containing very little antimony.

On the sorting level are the briquetting and roasting yards, briquetting being done by hand and roasting in heaps, both by contract. Below this level come the ore bins, and below these the ore beds, iron flux, charcoal and coke bins. This is the charge floor level. Limestone of a superior grade is quarried at one side of the plant, broken to size by women, and delivered to the charge floor through an inclined 10-in. pipe.

The smelting equipment consists of two 36-in. x 60-in. water-jacketed furnaces, of the usual type, one number four rotary blower, belt-connected to motor drive. The power for the plant was generated at a small hydroelectric installation in the valley, about seven miles distant from the plant. The native furnace men were very proficient in handling these ores, all of them having worked at the plant for years.

TABLE "A."

Class of material.	SiO_2 %	FeO %	CaO %	Al_2O_3 %	S %	Sb %	ZnO %	Pb %	Ag oz.
Socavón Ore, Crse.	46.1	14.9	6.5	6.5	1.8	7.8	9.8	2.0	32
" " Fns.	52.8	14.3	6.8	7.0	0.9	4.6	3.0	1.4	21
Sto. Entierro Ore	24.0	30.0	3.8	3.0	2.4	0.4	2.3	13.5	42
Calcines "A"	42.2	15.7	3.5	6.0	5.0	2.0	6.0	7.5	26
" "B"	20.8	28.0	3.7	3.5	8.0	0.3	8.5	12.8	31
Calcined Matte		64.0	2.8		5.1		2.0	8.5	50

As will be seen from the table the material available for smelting is not of the best. For the small furnaces the percentages of zinc, alumina and antimony are high, and the lead content low. The average analysis of the bed mixture was as follows: SiO_2, 30.5 per cent; FeO, 24.5 per cent; CaO, 7.0 per cent; Al_2O_3, 4.8 per cent; S, 3.7 per cent; ZnO, 4.6 per cent; Pb, 8.3 per cent; Sb, 3.6 per cent and Ag, 40 oz. The iron flux was very variable, from 10 per cent to 18 per cent SiO_2, 38 per cent to 48 per cent FeO, 0 per cent to 4 per cent Pb and 2 oz. to 10 oz. silver. The limestone was excellent, never falling below 52 per cent CaO, and carried about 2 oz. of silver. Welsh coke was used, but owing to the long transportation on pack animals the fines loss was excessive. The charcoal was exceptionally good, burned well and with practically no fines.

The typical daily furnace record, for 12 hours, slags produced, together with bullion and matte, and the extraction, are given in Table "B." Fourteen ounces air pressure at furnace was used, the tuyères were never bright and the lead in lead well always a dark red. The furnaces had to be very carefully watched, as the charge had a tendency to stick and bridge over above the tuyères. Accretions formed very quickly in the shaft and usually a campaign lasted at the most 32 days.

TABLE "B."

Furnace Mix.	Tons.	Production.	SiO_2 %	FeO %	CaO %	Al_2O_3 %	ZnO %	Sb %	Pb %	Ag oz.
Bedded Ore	16.6	Slag	34	32	18	9	3	1	1	3
Iron Flux	6.0	Matte							13	53
Limestone	3.8	Bullion						26	70	240
Slag	6.2									
Calc. Matte	1.0	Weight of Matte Produced, 1.634 tons.								
Charcoal	4.2	" " Bullion " 1.779 "								
Coke	1.3	Percentage Extraction, Lead, 83.7 per cent.								
Poor Lead	0.1	" " Silver, 82.3 per cent.								

Taking into consideration the refractory nature of the ore, the extreme volatility of antimony, the fuel used and the size of the furnaces, the slags are exceptionally low in both silver and lead. With suitable appliances for the recovery of dust and fume, and with an adequate forehearth, an excellent extraction of all values could have been obtained.

As in all isolated sections, the question of freights was a serious one. Coke laid down at the smeltery cost $21.75 per ton and charcoal $7. Ore, iron flux and limestone were laid down at a reasonable figure. Wherever possible work was done by contract, as mining, roasting, briquetting, tramming and bedding materials. The wages paid, both in the mine and smeltery, were very low, ranging from 12½ cents for boys and women per day to $1 for furnace foremen.

The cost of smelting per ton of ore was in the neighborhood of $12; this, however, did not include cost of marketing product, which was very expensive, as freight to the railroad amounted to $7.25 per ton.

The success of this plant during 1906 and 1907 was primarily due to the high prices of silver and lead (the selling contract called for antimony being paid for at the same price as lead). Aside from this fact, the extreme care taken when bedding the ores to have no fines and as little sulphides present as possible; to mix the ore mixture thoroughly before charging, and the screening of both charcoal and coke to insure the absence of breeze; finally, at no time would the furnaces be crowded. All of the foregoing facts contributed to the successful operation of the furnaces.

Metallurgically, the plant was interesting in many ways. The ore is an unusual combination; the bullion produced shows that antimony can be included as a collector of precious metals and presents but few difficulties from a smelting standpoint. It is rare that such a clean matte is produced, and contrary to expectations, this matte, except on very rare occasions, carried more than traces of antimony. It must also be borne in mind that smelting was carried on, comparatively speaking, with a cold-running furnace, the appearance of which, at its best, would have made any lead metallurgist, accustomed to bright tuyères and red-hot lead, very uneasy.

Campo Morado,
Gro., Mexico.

Zinc box precipitate may be refined with a solution of sulphurous acid as well as by sulphuric. The action is a little slower, but it has the advantage of being inexpensive, and is adapted to isolated places.

Recent Metallurgical and Chemical Patents

Iron and Steel.

Sintering Pulverous Iron Ore.—An interesting method of sintering pulverous iron ore, flue dust, magnetic concentrates, etc., and bringing them into suitable form for smelting in blast furnaces and for use in open-hearth furnaces in a single continuous operation has been patented by Mr. SIDNEY CORNELL, of Duquesne, Pa. When pulverous iron ore is heated to a temperature slightly above 2500° Fahr. part of the material is melted and the fused portion acts as a binder. This is probably due to the formation of ferrous silicates. For subsequent use of the clinker in the blast furnace or open hearth it is advantageous to return the ferrous compounds back to the ferric condition by subjecting the clinker while heated to oxidation. The process is carried out in a long heating chamber, the cross-section of which is shown in Fig. 1. From a hopper the material is charged into the pit 3 and carried along the whole chamber by means of a screw conveyor 12. The material is continuously forced into the pit until the pit is filled from end to end and the material is forced above the upper edges of the pit and passes out and over upon the upper faces of the curved walls 2, 2^1. The gas passing through the ducts 4, 4 and burners 6, 6, being ignited at the inner ends of the burners, heats the inside of the arched wall 1 to a high degree, and the mass of material being treated is heated directly and by the heat reflected from the arched wall. This induces a sintering of the outer portion of the material, which becomes incrusted with clinker. As more of the material is forced by the screw into and up through and out of the pit 3 it rises with a wave-like motion and breaks the crust or clinker formed upon the upper face of the pile of material, turning or folding over the broken pieces of the crust and exposing to the direct action of the heat the raw or unsintered sides, so that it becomes sintered on all sides. As the feeding process continues the broken clinker piles up upon the curved walls 2, 2^1, beyond the angle of repose (indicated by the dotted line 16), and then rolls down upon the tables 10, 11. When it reaches these tables the material consists of conglomerated iron oxides, but a large portion of which is in ferrous condition. On these tables the broken clinker remains sufficiently long for the air passing from below, through and over it into the chamber 5, to oxidize the ferrous oxide of the clinker into ferric condition, whereby the material becomes properly prepared and workable in the smelting furnace. When the clinker has laid long enough upon the tables to become sufficiently cool it is dumped into the car 17 below. The air passing over the clinker on the inclined tables cools the clinker, and is at the same time highly heated thereby, and while aiding the combustion of the gas passing from the burners also aids in the production of the higher temperature within the furnace. Any gases produced escape upward into a smokebox 8 and leave through the smokestack 9. In a single continuous operation the pulverous material is thus brought into masses of size and weight available for smelting in blast furnaces and for use in open-hearth furnaces, the ferrous compounds being simultaneously changed to the ferric condition. (962,006, June 21, 1910.)

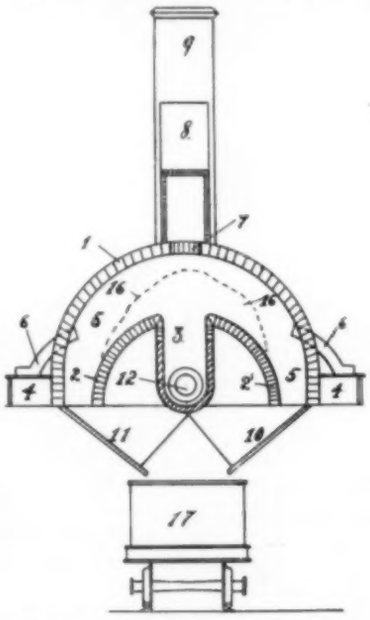


FIG. 1.—FURNACE FOR SINTERING PULVEROUS IRON ORE.

Steel Directly from Iron Ore.—Mr. FREDERICK T. SNYDER, of Oak Park, Ill., proposes the direct production of steel from iron ore in a single operation in an electric furnace by observing the following conditions: The minimum of carbon is used in the furnace charge required to secure reduction of the metal from the ore and produce steel. The steel collects in a crucible of basic refractory, such as dolomite, which is intended to take up small amounts of phosphorus and sulphur from the steel, but which cannot introduce carbon into the steel. The slag is confined by a fluid-cooled wall in contact with it so that the outlying portions of the slag bath are frozen, forming a lining of solid slag. The construction of the furnace is shown in Fig. 2, where 3 is the dolomite crucible, surmounted by the water jacket 5, which forms the furnace chamber; 6 are tap holes for the slag 9, while 7 is the tap hole for the steel 8, which is not allowed to accumulate in the crucible to such an extent that it would come into contact with the waterjackets. Carbon electrodes 10 (of which only one is shown) are suspended from the top into the molten slag, but are kept high enough so as not to come into contact with the molten steel in the crucible. The charge to be smelted is fed through the open top of the furnace onto the slag bath. The carbon electrodes are protected by a coating of lime, which prevents waste due to the absorption of oxygen from the oxide ore. (974,608, Nov. 1, 1910.)

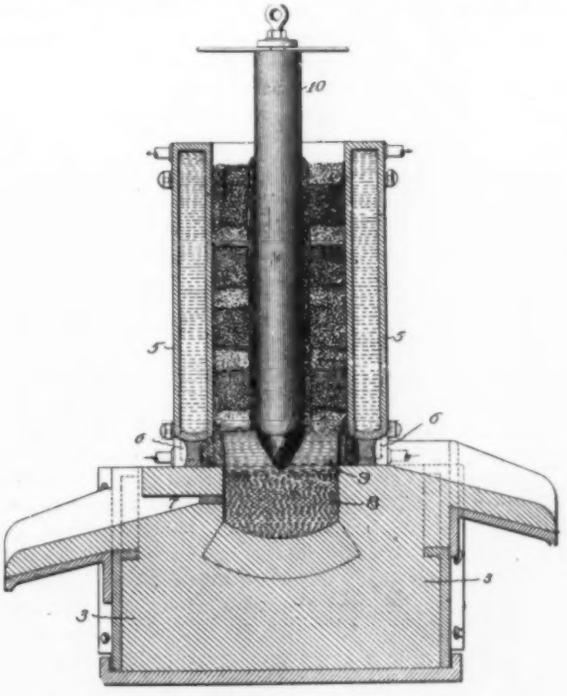


FIG. 2.—ELECTRIC STEEL FURNACE.

Steel from Ore in Electric Furnace.—An electric furnace of Mr. JOHN W. EVANS, of Belleville, Ont., Canada, is a combined arc and resistance furnace. Two sets of electrodes are provided; one set of opposite horizontal electrodes, the other of vertical electrodes suspended from the top. When operation is started the latter two electrodes are raised and not in

operation. An arc is started between the two horizontal electrodes and the upper portion of the charge is melted, producing a slag. The two horizontal electrodes are then withdrawn and the two vertical electrodes are lowered into the slag. The furnace now acts as a resistance furnace. If the material on one side cools off, an arc is produced between the vertical electrode and the horizontal electrode on this side. (960,862, June 7, 1910.)

To produce steel directly from iron ore in this furnace Mr. Evans mixes 10 per cent by weight of charcoal or tar with 90 per cent of iron ore and fluxing material and proceeds as described above. For the production of malleable iron a smaller quantity of carbon is required. The furnace is said to be "particularly adapted to treating ore containing titanium and vanadium, and by suitably regulating the quantity of limestone, sulphur and titanium can be slagged off and fine steel can be obtained, or if a hard tool-steel is desired when treating titaniferous iron ores less limestone may be used, and by regulating the quantity thereof any desired proportion of titanium can be retained in the steel." (960,863, June 7, 1910.)

Pure Metals from Impure Oxides and Sulphides.—Mr. J. W. Beckmann, of Niagara Falls, Ont., Canada, has devised an electrolytic furnace method for the production of pure metals, of which are mentioned iron, aluminium, chromium, titanium, tungsten, molybdenum, manganese and silicon, in general any metal whose oxide will form a salt with calcium. As applied to iron the process is as follows: Iron oxide is mixed with calcium oxide and heated in an electric furnace to produce a calcium salt of iron:

$$CaO + Fe_2O_3 = CaO\,Fe_2O_3.$$

Dibasic, tribasic or *n* — basic salts may also be produced. This salt is now used as a solvent for iron oxide, which is electrolyzed and yields iron according to the equation, $Fe_2O_3 = 2\,Fe + 3\,O$. Either direct current or alternating current may be used to obtain the bath in a state of fusion, while direct current is, of course, required for electrolysis. The fused $CaOFe_2O_3$ seems to act chiefly as a solvent for the iron oxide. Instead of the oxide, sulphide may be used. (973,336, Oct. 18, 1910.)

Pyrites Containing Zinc and Sulphur.—Concerning the treatment of waste pyrites containing zinc and sulphur remaining from the manufacture of sulphuric acid so as to recover zinc and ferrosilicon, see the patent of A. Wiens under Alloys.

Induction Furnace for Steel Refining.—See the patents of Ferranti and Gin under Electric Furnaces on pages 691 and 692.

Copper.

Recovery of Copper from Wet Process Liquors.—An improved method of obtaining copper from the liquors of wet processes treating roasted or unroasted ores, pyrites cinder, residues, ashes or scrap, or from mine water, has been patented by Messrs. J. H. Thwaites and S. J. Ralph, of Peterborough, England. The object of the invention is to obtain the copper practically free from iron and other metals contained in the liquor. Sulphite of magnesium is the means used by the inventors, although they find that the sulphites of calcium, barium and strontium give the same results, and with greater latitude in the operation. In the use of magnesium sulphite it must be restricted in quantity to a deficiency of the proportion necessary to precipitate the whole of the copper. It is preferred to heat the liquor with the sulphite and afterwards cool it, as in many cases the quantity of copper precipitated thus is greater per unit weight of sulphite. The precipitate can be treated in any manner desired, but as it is practically free from iron the authors suggest its use for the manufacture of copper sulphate, as this process would be cheaper and more rapid than the dissolution of copper in sulphuric acid. Cobalt, nickel and other metals contained in the liquor may be obtained later by precipitation as sulphides. (972,584, Oct. 11, 1910.)

A New Form of Copper Converter designed to avoid the usual destruction of the lining during the reduction of the matte to metal has been patented by Messrs. W. J. Murphy and A. E. Culley, of Butte, Mont. As shown in the upper diagram of Fig. 3, the converter is of the usual form except that it has the throat 19 at one end of the cylinder instead of in the center and is provided with a tap hole 20 and a spout so that the charge may be tapped. It is further provided with hoppers 22 and passages 21, so that fluxing material can be introduced and automatically spread on the bottom and sides

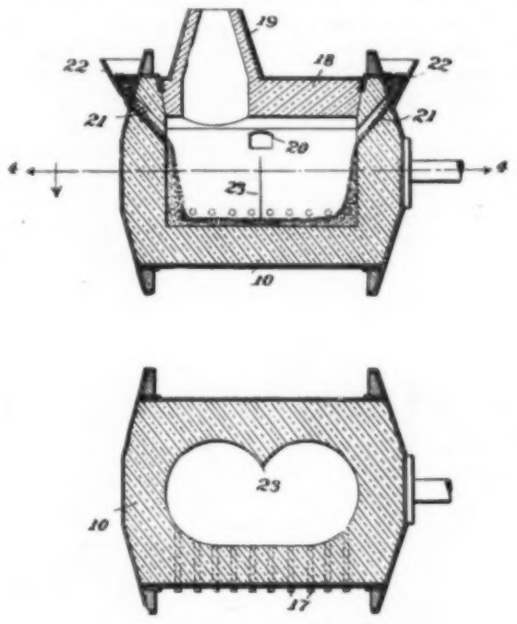


FIG. 3.—COPPER CONVERTER.

of the converter as shown. In the lower diagram, which is a sectional view on the line 4-4 of Fig. 1, is shown the form of the interior lining of the shell, having a rib 23 and the walls preferably rounded. This provides for the swirl of the charge in two directions as it is agitated by the blast from the tuyères 17, causing it to become thoroughly mixed with the charge of fluxing material. The converter could be used with either acid or basic lining and the nature of the flux suited to the matte. The inventors claim that such a converter may be used for a large number of charges without the necessity of relining. (972,564, Oct. 11, 1910.)

Handling Matte.—The usual manner of handling matte when it is not immediately converted to copper is to allow it to cool in molds, after which it is broken in pieces suitable for further smelting or transportation. In order to expedite this process, William D. Kilbourn, of Murray, Utah, has invented a process whereby the matte button is allowed to cool partly in a pan into which it is drawn from the furnace, then broken at a point on its edge and "bled" of its inner molten contents. This leaves a solidified shell which is easily broken by means of a stream of water directed upon it, and handling is thus reduced to a minimum. The smaller buttons formed by the bleeding of the larger are also easily broken by water as described. For the easier handling of the buttons loose anchor pins are placed in the pans when the molten matte is poured in them, so that the entire button can be picked up by the pin when the matte has solidified. (974,074, Oct. 25, 1910.)

Zinc.

Electric Zinc Smelting.—Mr. Woolsey McA. Johnson's work in electric zinc smelting has been known for a series of years. A recent patent granted to him forms apparently an important link in his inventions. In present practice of smelting zinc ores the ore is usually mixed with an excess of carbon above the amount necessary for reducton. There are two rea-

sons for this. Firstly, such a highly reductive charge insures the substantial reduction of carbon dioxide, aqueous vapor and other reactive products which would be capable of oxidizing zinc. Secondly, carbon is insured to be present in this way in the residue, whereby the latter is rendered pulverulent and infusible, any slag fusible at the temperature of reduction being absorbed or retained by the carbon; this is necessary with externally fired furnaces, because a fluid slag or matte in contact with externally fired fireclay retorts would quickly destroy

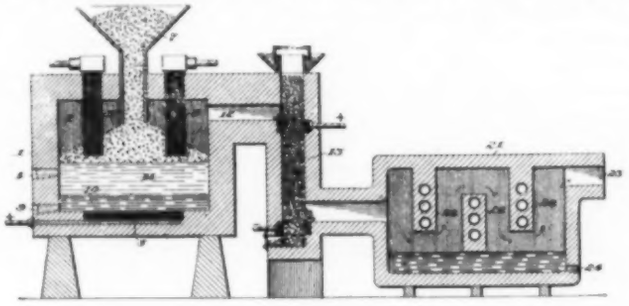


FIG. 4.—ELECTRIC ZINC FURNACE AND CONDENSER.

them. Mr. Johnson's method of operation is different in principle. He uses only such proportion of carbon as is necessary for reduction, producing thereby an easily fusible slag which may be tapped off, while the reduced lead, copper or iron accumulate beneath the slag and are withdrawn. He adds fresh portions of the charge continuously or at intervals, thus securing the advantages of continuous operation. Such operation logically requires the use of an electric furnace, because the production of fluid slags would be destructive to externally fired retorts, and any method other than retorting would require the introduction of such quantities of air in the form of a blast that it would be impossible to secure efficient condensation. The mineral components of the charge which enter into the slag should preferably be so proportioned as to yield a slag fusible at 1100° C. to 1175° C., such slag being superheated at the working temperature of the furnace, 1250°-1300° C., and therefore readily tapped. The gases derived from a charge of the character described are necessarily oxidizing in character, and this would, of course, interfere with condensation. For this reason the vapors are passed through a body of porous material which is artificially heated. This is best done by passing an electric current through the column of coke through which the vapors are passed whereby a more perfect reduction of the volatile products is secured than can be obtained by operating at a temperature of 1020° C. to 1150° C. upon a charge containing a large excess of carbon. The combination of apparatus used in Mr. Johnson's process is shown in Fig. 4, where 1 is an electric furnace with two upper carbon electrodes 3 of equal polarity and a carbon plate 5 in the bottom of opposite polarity. The charge is introduced through the hopper 7 into the furnace. 4 is the slag tap and 9 a tap for the removal of non-volatile metals (lead, copper, etc.) or matte. The zinc vapor passes through 12 over to the electrically heated column 13 of coke or coal, where the vapors are freed from carbon dioxide, aqueous vapor, etc. The purified zinc vapor then passes to the condenser 21, which is provided with water-cooled partitions 22 so as to direct the vapors in a tortuous path. The zinc is condensed in the hearth and tapped at 24. The non-condensing gases pass out at 23. As to the quintessence of his invention Mr. Johnson believes himself to be "the first to recognize that a continuous operation may be combined with a high degree of working efficiency by electrically smelting a charge adapted to yield a fusible slag and passing the vapors therefrom through a body of carbon at a reducing temperature and preferably at the temperature of maximum reductivity; that is to say, at a temperature at which carbon vapors are present." (964,268, July 12, 1910.)

Electric Zinc Smelting.—While well known by his former work in other metallurgical fields, Mr. HIRAM W. HIXON is a newcomer in electric zinc smelting. He employs a shaft furnace with a carbon bottom plate as one electrode and another carbon plate near the top as the other. The furnace is filled with zinc ore and carbon up to a point above the upper electrode. The charge acts as resistor. The top of the furnace is provided with a double hopper to prevent the zinc vapors from getting out and the air from coming in; an upper door may be opened for feeding in new charge, while a lower door is shut; the upper door is then closed and when the charge is sufficiently preheated the lower door is opened and the charge is fed into the furnace shaft. The zinc vapors are drawn off near the bottom of the shaft and passed into the condenser, while in the side of the condenser adjacent to the top of the tap hole a number of arcs are provided (probably in order to prevent clogging of the tap hole by blue powder). After the furnace is once started there should be no let-up in the vapor density, as the furnace is fed continuously at the top and the hot zinc vapors are forced to discharge at the bottom by keeping the furnace tightly closed. It is thus "possible to maintain the density of the zinc vapor at an approximately fixed point. This prevents the formation of the blue powder and results in the continuous production of the metallic zinc." (957,337, May 10, 1910.)

A modification of this process is described in another patent of Mr. Hixon. The chief new features are an exhaust pipe at the top of the furnace for pumping out the diluting gases, carbon dioxide and moisture, and of a tube inside of the furnace for drawing off the zinc vapor. This tube is made of a refractory material (magnesite or carborundum). It is placed in the center of the furnace and reaches from the bottom upward to a point below the upper electrode. It is closed at the top, but in its side walls inclined passages are provided through which the zinc vapors are intended to pass from the inside of the furnace into the tube and then to the condenser. (No zinc vapors can escape through the exit tube for carbon dioxide, etc., at the top, because at this point the temperature is too low for zinc to exist as vapor.) Otherwise the construction of the furnace is essentially the same as in the preceding patent. (957,058, May 3, 1910.)

Blast Furnace for Zinc-Iron Ores.—A patent of Mr. FREDERICK W. GORDON, of Philadelphia, is of considerable interest in view of Mr. Gordon's prominent connection with the large-scale experiments with the Lungwitz zinc process. Fig. 5 shows the blast furnace intended for the smelting of zinc-iron ores. 1 is the hearth, 2 the lower hottest zone, 3 the hopper, 4 the bell, 5 the downcomer, 6 the hot-blast tuyères and 7 the tap hole for the iron. As far as the reduction of iron from iron ore is concerned the furnace acts exactly like an ordinary blast furnace. Now, if the ore contains zinc, let us assume that line 8 indicates the region in the furnace column in which the temperature is high enough to vaporize reduced zinc, while line 9 may represent the colder region in which the temperature is low enough to condense the zinc. Then the zinc would vaporize at line 8, rise up and condense at 9, descend again to 8, vaporize and rise again and thus dance up and down between 8 and 9. This would be the precedure in an ordinary blast furnace. But Mr. Gordon provides a cooler 10 around this intercepting zone 20 between the lines 8 and 9 of the furnace. The details of construction are clearly shown in the diagrams. If the ports 11 and 15 are open, communication is established between the zone 20 in the furnace and the outer cooler 10. Gas with zinc vapor passes from the inside of the furnace through the ports 11 into the outside cooler, where it is forced to rise through 12, descend through 13 and rise again through 14. Condensed liquid zinc is withdrawn through the tap hole 17 (peep holes being provided at 18 and 19). The non-condensed vapors pass through the ports 15 back into the furnace. The vapors "enter the cooler at a temperature approximating that of vaporization under the pressure within the furnace, assumed at 17 lb. absolute or 2.3 lb. gage per square inch, and

leave the cooler at a temperature about that of the solidification of zinc." In continuous operation of the furnace the withdrawal of molten zinc from the cooler should correspond with practical precision with the quantity of zinc contained in the

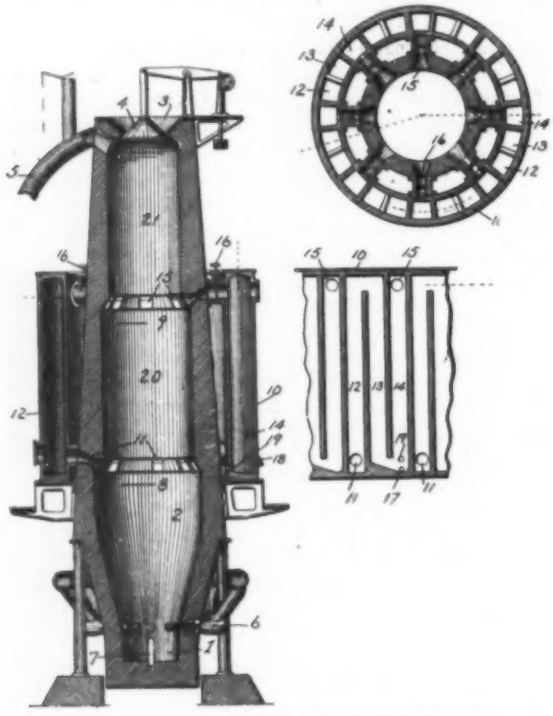


FIG. 5.—BLAST FURNACES FOR ZINC-IRON ORE.

new additions of ore made to the charge at regular intervals. The upper zone 21 of the furnace acts simply as a preliminary heating chamber to utilize the heat of the waste gases. (974,257, Nov. 1, 1910.)

Direct Treatment of Blende.—A new improvement in the treatment of zinc sulphide is suggested by ADOLPHE DESGRAZ, of Hanover, Germany, in a patent recently granted to him and assigned to Imbert Process Company, of New York. The author recites the former methods of reducing zinc ores, comprising roasting and reduction with carbon, or direct reduction of the sulphides by fluidifying them and adding a precipitating substance, such as iron, to precipitate the zinc. He states that various vehicles have been suggested for fluidifying the blende, such as a mixture of lime, oxide of iron and manganese; peroxide of iron and monosulphide of iron; or, peroxide of iron alone. It is pointed out, however, that all of these substances have a value in themselves and would cost something for transportation, etc. The inventor shows the need of a fluidifying material that is plentiful, cheap and otherwise suitable. Such a substance he finds in ordinary slag from lead furnaces. Lead slag, he states, is an excellent dissolvent for blende, i. e., allows of its fluidification at low temperatures, is cheap, contains some zinc, and is generally suited to the purpose indicated. He suggests the following treatment: Melting the slag and blende in a suitable closed furnace, from which the mixture can be tapped to a precipitating vessel where cast iron is used to precipitate the zinc from the mass. (975,217, Nov. 8, 1910.)

Improved Chloridizing Process.—For the treatment of mixed oxidized and sulphide ores of zinc, copper, etc., CHARLES E. BAKER, of Chicago, Ill., finds that a mixture of hydrochloric acid gas and chlorine gas gives better results than either reagent separately. He prefers that the chlorine should be proportioned to the quantity of sulphides in the ore, while the hydrochloric acid gas is used in proportion to the oxides and carbonates present. A typical reaction is given as follows:

$$2\ ZnO + ZnS + 4\ HCl + Cl_2 + O_2 = 3\ ZnCl_2 + 2\ H_2O + SO_2$$

It is merely necessary that the ore be properly mixed with the reagents in a revolving drum or other suitable vessel. The author claims an efficient chlorination with absence of agglomeration of the ore. (972,149, Oct. 11, 1910.)

Zinc Oxide.—A patent of Messrs. WILLIAM SIMM and HENRY SIMM, of Portico, near Prescot, England, describes details of mechanical construction of apparatus for rapidly producing electrically zinc oxide from zinc ores such as calamins and metallic substances containing zinc. The features of the apparatus are sufficiently indicated by the second claim which relates to "the combination of a condensing chamber having suitable baffles, a hood connected therewith, and a suction fan for drawing the fumes into the hood and discharging them into the condenser, with an electric furnace removably located beneath the hood, a space being provided between the bottom of the hood and the top of the furnace whereby air for the supplying of oxygen is freely admitted on a level with the heating material carried by the furnace." (946,688, Jan. 18, 1910.)

Alloys.

Antifriction Alloy.—An inexpensive alloy, affording slight friction and sufficiently hard to reduce abrasion to a minimum, has been patented by JOSEPH R. STRATTON, of New Brunswick, Canada. His discovery lies in the fact that the addition of spelter to a combination of other metals produces an excellent antifriction metal suited for railway and heavy service. The formula preferred by the inventor is as follows:

	Pound.	Ounce.	Per Cent.
Copper	0	8	1.7
Tin	1	12	5.8
Lead	3	0	9.8
Spelter	25	0	82.0
Antimony	0	3	0.7
Total	30	7	100.0

The metals are melted together and thoroughly mixed in any preferred manner. (972,786, Oct. 11, 1910.)

Copper Alloy.—A new chromium alloy of copper compounded with nickel and other metals has been invented and patented by Messrs. JAMES NAULTY and JOHN SCANLIN, of Philadelphia, Pa. The alloy is of great tensile strength and approximates the hardness of ordinary steel, so that edged tools can be made from it. The alloy is made by melting in a crucible 5 lb. of chromium, with 5 lb. of aluminium of commerce used as flux. When thoroughly melted, after about an hour, 110 lb. of copper are added to the mass and melted. This requires about 30 minutes, after which 25 lb. of nickel are added and the melting continued for another hour. Finally 20 lb. of zinc are added. The ratio of chromium to copper and nickel to copper may be varied, but the order in which the metals are admixed and the use of aluminium as a flux are essential. The alloy has a slightly reddish tinge and gives a tensile strength of 78,950 lb. (974,419, Nov. 1, 1910.)

Ferrosilicon from Sulphur-Containing Iron Ores.—When using sulphur ores for the production of ferrosilicon a product containing a large amount of sulphur is obtained even in electric furnaces when the content of silicon is relatively low. But Mr. ARNOLD WIENS, of the Elektrochemische Werke, of Berlin-Bitterfeld, in Bitterfeld, Germany, has found that by increasing the silicon content by the addition of sufficient silica to the furnace charge the sulphur content can be greatly reduced. The reduction must, of course, be carried out in an electric furnace. For example, when obtaining a ferrosilicon

with 6.4% silicon, 3.1% sulphur is present;
" 8.5% " 2.8% " " "
" 12.0% " 2.4% " " "
" 22.0% " 0.12% " " "
" 31.4% " no sulphur is present.

Therefore, if the ferrosilicon contains at least 20 per cent Si an entirely marketable alloy is obtained. "If the ore contains other metals besides iron, such, for example, as zinc, which is very often the case in common iron pyrites, it is essential that the material should be first roasted; substances are then added thereto, with which the remaining sulphur that is set free combines without being additionally driven off either by carbon or by silicic acid. By these means the zinc is recovered in receivers in the metallic form or as oxide, but without sulphur. Lime is generally used as such a sudphur-absorbing substance." The waste pyrites containing zinc and sulphur which remain from the manufacture of sulphuric acid can be treated in this way to recover zinc and ferrosilicon. "Zinc ores that are only reduced with difficulty can be added to the iron ores ,as in principle it is immaterial whether oxidized or sulphurized ore predominates. Further, it is of the greatest importance in the usefulness of the process that since the electric furnace is entirely closed a very pure carbonic acid gas is obtained under the circumstances." (973,732, Oct. 25, 1910.)

Furnace Slag.

Utilizing the Heat of Slag.—The generation of low-pressure steam from hot furnace slag is the basis of a patent recently granted to CLAUD VAUTIN, of London, England. An extended explanation of this system was given in our issue of May, 1910, page 297. It will be observed from Fig. 6, however, that the system involves the use of a slag hopper *C*, with a trap *C'*, from which molten slag flows into water contained in a tank *D*. As it enters the tank it is struck with a stream of water as shown at *M'*, the water being drawn from the tank

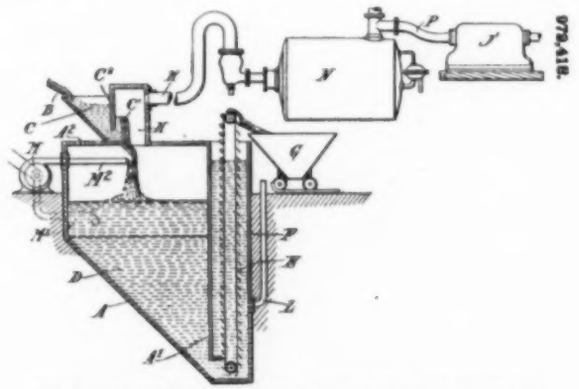


FIG. 6.—APPARATUS FOR UTILIZING THE HEAT OF SLAG.

through the pump *M*. The granulated slag which has given up its heat to the water falls to the bottom of the tank and is elevated through the elevator *A'* into a pot *G*. The steam generated is led past the inflowing stream of hot slag, and thence through an accumulator *N* to a low-pressure turbine *J*. (972,418, Oct. 11, 1910.)

Electric Furnaces.

Mixing by Induction.—Mr. S. Z. DE FERRANTI, of Grindleford Bridge, near Sheffield, England, who is one of the original pioneers of the electric induction furnace, has recently devised several modifications of the induction furnace and especially worked out methods of stirring and agitation by induction. One of his designs is an induction furnace with an oval annular melting chamber, in which the metal is heated by induction currents produced in the same way as in the ordinary induction furnace. The new feature is that at a certain part of this annular melting chamber special windings are provided with operative portions normal to the free surface of the metal, with iron masses, the arrangement being exactly analogous to the stator of an induction motor. These produce a rotary magnetic field which, acting on the metal in the melting chamber in the same way as on the rotor in an induction motor, produces a movement or rotation of the metal and thereby thorough stirring and mixing. Of course, the ordinary induction furnace may be left unchanged and the molten metal, after having been sufficiently heated, run off into a special mixing chamber, which is surrounded with windings to produce a rotary magnetic field. (962,013, June 21, 1910.)

Combined Open-Hearth and Electric Furnace.—A further development of the same idea by Mr. Ferranti is shown in Fig. 7. which represents a horizontal and vertical cross-section of a Siemens regenerative furnace with an electrical arrangement for mixing and circulating. The furnace *a* proper is of circular form, with three auxiliary channels *b* of comparatively small section in communication with the central portion and surrounding the three vertical limbs of the magnet core. On

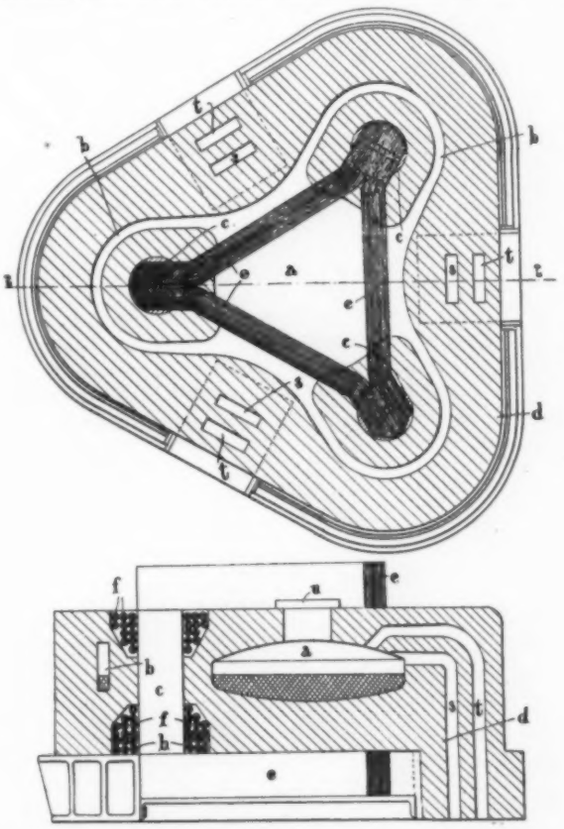


FIG. 7.—COMBINED OPEN HEARTH AND ELECTRIC FURNACE.

the limbs *c* of the three-phase arrangement coils *f* are wound above and below the furnace bath in the corners of the encircling magnets so that the turns of the coils are roughly in the form of a double cone with the furnace bath disposed in part in the angular spaces between the cones. Additional coils *h* are provided in the lower parts of the magnet limbs, so that the unsymmetrical field produced may rotate or mix the material acted upon. The central body of the bath is violently rotated by the leakage field, thus inducing a flow around the subsidiary channels *b*, which are intensely heated electrically. The entire mass is thus caused to pass continually through the channels *b*. When the metal is rotated in the central compartment a hollow forms in the center by centrifugal force and this may be made use of for drawing off the spent slag by means of a duct. The usual regenerators are indicated at *s* and *t*.

Among the various applications described is an intermittent acid process with separate complete heats for working iron with very low phosphorus and sulphur and high in carbon, manganese and silicon. The bath of the electric furnace is first filled with a charge of molten pit iron taken from the mixer or from the blast furnace direct. The bath is then electrically heated and mixed. Iron ore is added and the blast is started from the nozzles above the bath so as to "drive in oxide which is churned up with the metal by the blast and electric mixing.

The oxygen in the blast at the same time acts on the metal. The heat produced by the chemical reaction and the electric current is very great, and as the mixing is most complete the impurities are rapidly oxidized. The metal being spun around by the leakage field piles up on the sides and its surface assumes a dish form. In this dish so formed the reaction takes place, thus saving the walls of the furnace from the action of the slag. As soon as CO is given off, the blast may consist of air only so as to oxidize it to CO_2. The metal may have carbon added, for example, in the form of spiegel. Instead of pouring immediately, as in the Bessemer process, the metal may be kept in the furnace to be further heated and mixed and only poured after sufficient treatment, such as the addition of an alloy. The effect of this operation of heating and rotating is to free the metal from occluded gas, burned particles and slag. Slag may be used in this process to any desired extent." The treatment of iron high in phosphorus is also described. (969,261, Sept. 6, 1910.)

Induction Furnace Design.—Mr. Gustave Gin, of Paris, France, has designed an electric steel furnace which, in the form of his long-extended narrow channel, resembles much his older furnace with steel blocks as electrodes which attracted some interest when electric steel refining first came up for general discussion (our Vol. II, page 20). Mr. Gin has now given up his steel-block electrodes and heats the channels by induction. He still employs elongated, endless and parallel channels of uniform cross-section, with extended portions folded back into parallelism with each other. "The energizing magnet is so disposed relatively to said parallel portions that parts of latter lie relatively outside of the magnetic influence and so close to each other as to largely reduce self-induction and reactance, the distance of said parts from said magnet being much greater than the diameter of the horizontal cross-section of said magnet taken in the plane of said crucible." (970,700, Sept. 20, 1910.)

A patent of Mr. Paul Gredt refers to some details of construction of an oscillatory induction furnace with annular hearth and especially to methods of charging it by means of conveying worms which revolve in tubes surrounded by heating jackets, so as to preheat the fresh charge. Tapping holes are placed at different levels for drawing off either the slag or the molten metal. (950,911, March 1, 1910.)

Aluminium Furnace.—Mr. Heinrich F. D. Schwahn, of Belleville, Ill., has devised an electric furnace for the smelting of metals, and especially aluminium (his aluminium process having been noted in our September issue, page 547). The smelting chamber is a ring-formed retort and in the center of the ring the step-down transformer which supplies the electrical power is placed. Two sets of electrodes are provided; one set in the bottom of the retort, consisting of two plates made of powdered carbon and magnesite and connected to the secondary of the transformer. The other two electrodes are water-cooled copper or aluminium plates inserted in the sides of the retort, near its top. They are connected either to a source of direct current or also to the secondary of the transformer. Graphite pipes are inserted in the walls for the introduction of reducing or fluxing gases into the furnace. Above the annular furnace chamber a gas-hood is provided formed of two concentrical cones; the space between these two cones is used for carrying off the spent gases and for introducing and preheating fresh charge. The outside air is excluded by this means from the furnace. (962,532, June 28, 1910.)

Electrolytic Furnace.—Mr. H. A. Danne, of Clarksburg, W. Va., employs a cylindrical crucible which is closed on the top by an annular cover with an open hole in the center. Upon this annular cover, so as to fit into the central hole, a shaft is placed which receives the charge from above, revolving blades being provided for stirring. At the bottom of this shaft (hence at the top of the crucible) the positive electrode is provided, formed of powdered carbon cemented together and suitably reinforced. This electrode increases in diameter downwardly in step-by-step fashion (or in a frusto-cone shape). This positive electrode is surrounded by groups of negative electrodes suspended from the annular part of the crucible cover; each group is disposed in a different horizontal plane corresponding to the step arrangement of the opposite electrode. The furnace is intended for continuous operation (for instance, in the manufacture of aluminium). When the charge is decomposed between the electrodes it passes away and new charge is fed between the electrodes. The fresh charge descending from above, is thoroughly preheated. (958,855, May 24, 1910.)

Calcium Carbide.—A patent of Mr. James E. Hewes, of Plattsburg, N. Y., refers to an electric arc furnace for the production of calcium carbide. Below two oppositely directed horizontal water-jacketed electrodes and at a slightly lower level, a traveling sectional hearth is employed for supporting the carbon and lime to be smelted. This hearth consists of a series of carriages with bottoms and sides, but no ends, and abutted end to end to form a long trough-like structure. As this hearth travels the carriages are added at one end and removed with the carbide at the other end. (969,622, Sept. 6, 1910.)

A patent of Mr. Herman L. Hartenstein, of Constantine, Mich., refers to details of construction of tap-hole for calcium carbide, the object being to prevent the tap-hole from being clogged. (971,445, Sept. 27, 1910.)

Cyanamide Furnace.—In our September issue, p. 548, we described the process of Messrs. E. H. Meyer and J. M. A. Stillesen, of Niagara Falls, Ont., Can., for the production of calcium carbide and cyanamide. Details of construction of their revolving inclined cyanamide furnace, which is charged from the bottom and in which the relative distance between the electrodes is automatically controlled according to the load, are given in their patent 955,655 of April 19, 1910.

Electric Glass Furnace.—Various forms of electric furnaces for the continuous manufacture of glass are described in two patents of Mr. Marius Sauvageon, of Columbes, France. All of them have in common the combination of a relatively small smelting and refining chamber with one or two larger working chambers. The smelting chamber contains two electrodes, either at the two ends or one above the other. The furnace acts as a simple resistance furnace, the molten glass being the resistor. Fresh raw material, frit, is continually fed onto the top of the molten glass in the melting chamber, so that the electrically produced heat in the molten glass is employed to melt down the frit. The refined glass passes through a passageway into the larger working chamber, where it is cooled to a suitable working temperature. (972,778 and 972,779, Oct. 11, 1910.)

Manganates.—Mr. F. J. Machalske patents a process of producing alkali manganates by heating a mixture of the alkali haloid and manganese dioxide in presence of air in an electric furnace. For instance, by heating two parts of sodium chloride and one part of manganese dioxide he produces one part of disodium manganate and two parts of free chlorine. The chlorine may be collected separately or it may be used for converting the manganate into permanganate according to the equation $2Na_2MnO_4 + 2Cl = 2NaCl + Na_2MnO_4$. The sodium manganate may be treated with water, three parts of the manganate and two parts of water giving one part of manganese dioxide, one part of sodium permanganate and four parts of sodium hydroxide. The manganese dioxide may be returned to the electric furnace as part of a new charge. The sodium permanganate may be converted into potassium permanganate by causing it to react with potassium chloride. (956,120, April 26, 1910.)

Electric-Arc Furnace.—A kind of electric-arc reverberatory furnace is the subject of a patent of Mr. J. F. Shawhan, of Dayton, Ohio. A current of air is passed over an electric

arc formed between a set of electrodes, and the heated air is pressed by air pressure into a working chamber which may be the hearth chamber of a reduction furnace or of a reverberatory furnace. The electrodes are placed outside of the furnace and the arc is formed in a twyer sleeve of refractory material opening into the working chamber. The air is passed through this sleeve and is heated by the arc and by contact with the muffle walls. The air blast is forced at such a pressure that the arc is drawn out and deflected into the working chamber. This blast of air performs a threefold function, it carries the heat necessary for reduction or refining, it furnishes the oxygen and it blows up the slag. For drawing out the arc electromagnetic means may also be used. When the arc is drawn out it is maintained at the extremities of the electrodes so they keep their rounded form and burn off evenly. (951,458, March 8, 1910.)

Revolving Arc Furnace.—A patent of Mr. CHARLES A. WEEKS, of Philadelphia, Pa., refers to details of mechanical construction of an arc furnace with a ring-formed furnace body and a cylindrical furnace chamber in the center. One or several pairs of electrodes are provided in the center and the whole furnace revolves so that its lining is worn out uniformly. (949,511, Feb. 15, 1910.)

Arc Furnace.—A batch of patents, granted to Mr. J. H. REID, of Newark, N. J., relates of various constructions of arc furnaces. In one type a fixed bottom electrode is provided in form of a thick carbon rod. Above this and inclined to both sides are adjustable electrodes adopted to form a hopper to receive the ore. Arcs are produced between the inclined side electrodes and the bottom electrode and act on the ore. The position of the side electrodes is automatically regulated so as to permit "only ore in a more or less molten state to pass through." It then drops directly into the quenching tank (947,849, Feb. 1, 1910). In another type of furnace the charge, after having been subjected to the action of the electric arcs, is then subjected to the action of gases passed into the furnace through pipes inserted in the walls below the electrodes; these gases are sucked in by exhausting the upper part of the furnace. (950,116, Feb. 22, 1910.) The product of the furnace process, together with the slag, is then withdrawn into another cylindrical vessel which is heated by means of electrodes inserted through the walls and which is rotated, the object being to separate "the metals of different specific gravities by rotation of the containing vessel." The vessel is at the same time exhausted to remove the metals which volatilize. (950,028, Feb. 22, 1910.) Instead of passing the charge through a single furnace chamber as described above, several furnace chambers may be arranged on top of each other so that the charge, having passed through the action of one set of electrodes in one chamber, is subjected again to the action of another set of electrodes in the next lower chamber and so on. (968,079, Aug. 23, 1910.) Provision is made for independently and automatically regulating the movement of any electrode according to the current passing through it, while the movement of all the electrodes is effected from a single source of power. (968,603, Aug. 30, 1910.)

Electrode Support.—Mr. ISAAC R. EDMANDS, of the Electro-Metallurgical Company, of Niagara Falls, has patented details of adjustably supporting the depending electrodes used in electric furnaces, the electrode being counterbalanced by counterweights by means of flexible conductor ropes. The mechanical arrangement selected permits of a wide range of adjustment of the electrode, while requiring but little head room above the furnace. (959,486, May 31, 1910.)

Various Electric Furnace Processes are also described above under Iron and Steel, Zinc and Alloys.

Electrolytic Processes.

Treatment of Copper-Nickel-Iron Alloys.—An electrolytic process for the treatment of nickel-copper or nickel-copper-iron alloys for the production of copper and a nickel-iron alloy has been devised by Mr. P. F. COWING, of New York City, the process being shown diagrammatically in Fig. 8. The original nickel-copper-iron alloy is used as anode 1 in the diaphragm cell *A*, a warm concentrated neutral aqueous solution of nickel chloride being used as the initial electrolyte in the anode compartment. By anodic action the metals of the anode *A* pass gradually into solution, which then contains nickel chloride, ferrous chloride and cuprous chloride. This anodic solution is withdrawn into a second electrolytic cell *B* in which selective deposition is carried on, copper alone being deposited on the cathode 7 by maintaining a sufficiently low voltage. The anode 5 consists of nickel or iron or both. The solution is heated and agitated by means of the stirrers 7. Most of the copper is here deposited from the solution which now contains essentially nickel and ferrous chlorides and some cuprous chloride. This solution is passed into the tumbling barrel *C*, where the rest of the copper is removed by cementation on iron scrap 11. The solution, which now contains nickel and ferrous chlorides, is run into the settling tank *D* and then lifted by means of the pump 13 into the heating chamber *E*, from where it is passed

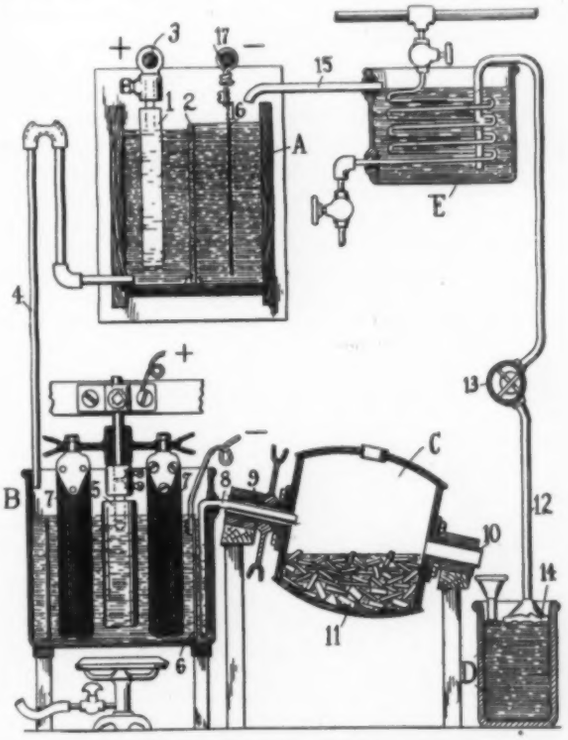


FIG. 8.—TREATMENT OF NICKEL-COPPER.

into the cathode compartment of cell *A*, where an iron-nickel alloy is deposited on the cathode 16. The solution in the cathode compartment is permitted to flow over the diaphragm 2 into the anode compartment, thereby completing the circuit of flow. Throughout the process the iron and copper salts are formed and should be maintained as far as possible in their reduced condition as ferrous and cuprous salts. As no insoluble anodes are used the electrolyte remains at all times electrochemically balanced. The anodes should contain as small a percentage of sulphur as possible to keep down the formation of sulphates, and these, if formed, may be removed by the occasional addition to the solution of barium chloride. (969,773, Sept. 13, 1910.)

Production of Ferric Salts.—A recent patent of Mr. ANSON G. BETTS, of Troy, N. Y., refers to the production of ferric salts by the electrolysis of ferrous salts of polybasic acids, for example, ferrous sulphate and ferrous fluosilicate. One of the difficulties encountered is polarization at the anodes and destruction of the carbon anodes. This may be overcome

by employing a large relative movement between anode and electrolyte. But without such movement it may also be overcome simply by electrolyzing at a high temperature. For instance, a solution containing 2.8 grams iron as ferric sulphate, 1.6 grams iron as ferrous sulphate, 1.75 grams copper as copper sulphate and about 4 grams free sulphuric acid per 100 c.c. can be electrolyzed at a temperature of 86° C., with a graphite anode, at an anode current density of 11 amp per square foot, without showing more than slight extra polarization beyond that necessary for the formation of ferric sulphate. The same solution at a temperature of 70° C., with a current density of 8.5 amp per square foot, shows an extra polarization of half a volt or over, which is still greater at lower temperatures. With a current density of 14.5 amp per square foot there is an extra polarization of over half a volt at 84° C., showing the necessity of using a still higher temperature at the higher current density. By circulating the electrolyte, or moving the anode, a lower temperature may be used In fact, especially good results are obtained by empoying considerable relative motion and some heat. (969,833, Sept 13, 1910.)

Hypochlorite Cell.—While in the production of caustic soda and chlorine by electrolysis of common salt solutions the object must be to keep the chlorine produced at the anode away from the caustic at the cathode, the opposite is required in the electrolytic production of hypochlorite (bleaching liquor) solutions from common salt. Here the anodic chlorine must be made to react with the cathodic caustic. Mr. PAUL MCDORMAN, of the National Laundry Machinery Company, of Dayton, Ohio, has devised a cell in form of a rectangular box through which the liquid (sodium chloride solution) is forced to pass in a circuitous path. An anode is provided at one end and a cathode at the other end. Between them there is a series of electrodes acting as bipolar electrodes; that is, they are not connected to the external circuit, but the electric current passes through them from the liquid and into the liquid so that one face of each of these plates acts as anode, the other as cathode. These intermediate bipolar electrodes do not fill the whole cross-section of the cell, space being left above and below each electrode. These spaces are filled with glass plates, one being inserted above and below each electrode. In order to permit the solution to pass, recesses are formed in the glass plate above the first intermediate electrode, in the glass plate below the second electrode, in the glass plate above the third electrode, and so on. The solution is therefore forced to pass through the recess over the first electrode, then through the recess below the second electrode, and so on. (968,492, Aug. 23, 1910.)

Electrolytic Cell.—An electrolytic cell in which special provision is made for circulation of the electrolyte and for artificially cooling it is patented by Messrs. GILBERT C. LANDIS, of Wadsworth, Ohio, and Jacob H. Smith, of Rittman, Ohio (assigned to Ohio Salt Company). The cell contains a series of electrodes and the electrolyte is passed from one end to the other, passing over the top of the first electrode, below the bottom of the second, over the top of the third, and so on. Between each two electrodes a glass coil is provided through which cooling water is passed. (974,576, Nov. 1, 1910.)

Chemistry.

Synthesis of Ammonia.—Mr. H. S. Blackmore, of Mount Vernon, N. Y., proposes to make ammonia as follows: A vessel containing mercury is subjected to agitation and then nitrogen and hydrogen gas is introduced into the mercury and thereby caused to unite and form ammonia NH_3, some of which, together with a portion of the hydrogen, unites with the mercury producing ammonium amalgam, which may be finally recovered as ammonia by heating the amalgam so formed. It is recommended to heat the hydrogen and nitrogen gases. Instead of passing the nitrogen and hydrogen through agitated mercury the mercury may be atomized with a blast of compressed nitrogen and hydrogen. It is recommended to use mercury containing calcium, as the latter facilitates the union of nitrogen and hydrogen by absorbing the hydrogen at ordinary temperatures, transiently forming intermediately calcium hydride, which is broken up into ammonia by action of the nitrogen. If methane or natural gas is used instead of hydrogen, methylammonias are produced, such as methylamin, dimethylamin, trimethylamin, or methyl ammonium compounds. By employing such substances as methane or natural gas, nitrogen may be fixed in such a manner as to form compounds or salts with suitable acids, which are valuable as fertilizing agents. By passing a current of electricity through the mercury during agitation with the nitrogen and hydrogen "union and formation of ammonia is further facilitated by the numerous electric sparks produced by the rapid completing and breaking of the electric circuit throughout the mass during agitation." (974,633, Nov. 1, 1910; modifications are described in two other patents of Blackmore, 974,741 and 974,742, Nov. 1, 1910.)

Cleaning Furnace Gases.

Dust Arrester.—A device for use primarily in the manufacture of sulphuric acid has recently been patented by JOHN HICKEY, of Buffalo, N. Y. The invention relates to dust arresters of the class comprising a group of baffles or deflectors through which the dust laden gas is caused to flow and which intercepts the dust. In Fig. 9, which is a fragmentary elevation of the device, *A* represents a chamber in which the collector *B* is located. The gas enters at *a*, coming from the furnace, and has an exit at *a'*, passing to the chambers. Spaced baffles or deflectors *D* are supported in the chamber. They are con-

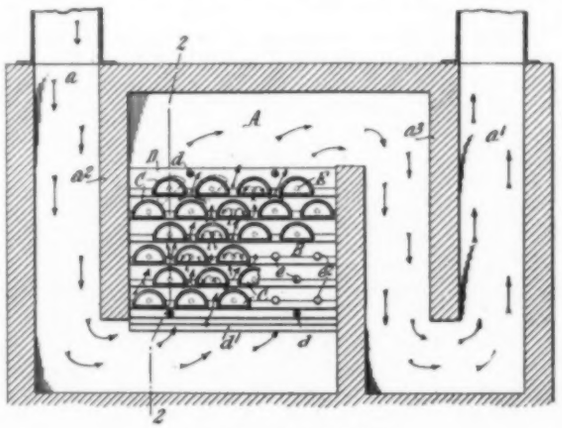


FIG. 9.—DUST ARRESTER.

siderably wider than the intervening spaces, so that the rising gases impinge on the concave surfaces of the baffles and are deflected in two directions against the convex upper surfaces of the two baffles below. The dust is thus allowed to settle on the convex surface, from which it occasionally falls onto the baffles below and thus to the bottom of the chamber. The baffles may be of an inverted V shape instead of semicircular as shown. Provision is made for the rocking of a tier of baffles should it become necessary to dislodge dust from them. (972,534, Oct. 11, 1910.)

Gas Washing Machine.—For cleaning producer or blast furnace gas, removing dust, tar and other impurities a special machine has been patented by WILLIS H. CARRIER, of Buffalo, N. Y. The patent is assigned to the Buffalo Forge Company, Buffalo. The machine comprises a casing enclosing three impellers or fans, each rotating within a chamber. The impellers rotate together and may be regarded as parts of a single compound impeller. As the impellers rotate rapidly the impure gas is drawn through an intake where it is wetted by a spray. Passing into the first impeller chamber the gas is set in rapid motion, and the heavier particles of tar, etc., are thrown out into an impact chamber at the periphery of the

impeller. From the impact chamber the gas passes into the spray chamber, where it is again set in motion by the second impeller. Here it is caused to meet sprays of water which further wash the gas, while the rapidly revolving impeller throws the condensed impurities to the periphery of the impeller, where they are removed. Passing through the second and third impellers the gas is dried and flows out of the machine. In its passage out, however, it automatically operates a pressure device so that the machine delivers gas at practically constant pressure. In order to prevent the condensation of light tars which have remained gaseous during the passage through the machine the gas is passed through a superheater before going to the gas engines. This prevents the condensation of these products in the gas engine. (973,626, Oct. 25, 1910.)

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

Refractory Low-Grade Ore.—Rather unusual means taken to recover gold from a refractory ore are described by Mr. R. H. KENDALL in the October *Bulletin* (No. 75) of the Inst. of Min. and Met. The ore of the Ouro Preto mine, in Brazil, is composed of quartz, tourmaline, arsenical and iron pyrites, some bismuth and free gold up to nearly 60 per cent of the fire assay. Of the latter, about 5 per cent will remain on a 40-mesh screen, 45 per cent on 100-mesh and 50 per cent will pass 100-mesh screen. Amalgamation was originally tried, but owing to the bismuth present the process was not successful, and the old-fashioned practice of recovering free gold on blankets was installed, supplemented by the cyanide process. The efficiency of this system is excellent, showing a total recovery of about 92 per cent on ore worth $7 to $8 per ton.

There are two mills of practically the same design and equipment, located one below the other on the hillside. The equipment of each comprises Blake crushers, 40 stamps of 750 lb. weight, blanket tables, Frue vanners, classifiers and cyanide tanks. Water-power is used and the stamps are driven by double vortex turbines at a speed of 96 drops per minute, with an 8-in. drop and discharge of about 4 in. The screen is 30-mesh and the average tonnage per stamp is 3.2 tons, but owing to the fact that the mill is usually ahead of the mine, only 2.8 tons are treated per stamp per day.

Some coarse gold, amounting to about 6 per cent of the assay value, is recovered from the mortar boxes. The pulp from the stamps is run over blanket tables set on a gradient of 1½ in. per foot. Each blanket is 18 in. wide and 4 ft. long. There are four blankets to each set of five stamps, or 32 in all. They are changed two or three times an hour, the work of changing all blankets requiring about 10 minutes. The blanket concentrate contains practically all the free gold in the ore, and is subjected to successive treatment on other blanket tables and in bateas whereby the final residue is enriched to about 800 fine. The ratio of concentration effected before gold is first recovered is 10,000 into 1.

The pulp flowing from the blankets in front of the stamps is treated on Frue vanners, but before flowing onto the belts it is treated on shaking blankets fixed to the distributors of the tables. A rich concentrate is obtained from these blankets, amounting to 6-7 tons per month and assaying about 16 oz. gold per ton. The vanners produce about 300 tons per month of concentrate assaying about 1.2 oz. gold per ton.

The vanner concentrate, together with the residue from the reconcentration of the first blanket concentrate, is treated in the concentrate cyanide plant. The vanner tailing, assaying about $1.20 per ton, is classified into sand and slime, and each product is treated by cyanide. There are thus four distinct products treated by cyanide, viz., the *rich concentrate* from the shaking blankets on the vanner distributors, to which is added the residue from the last regular blanket concentration; the *ordinary concentrate*, which is made up of the vanner concentrate and the residue from the first regular blanket concentration; the *sand* from the vanner tailing, and the *slime* from the same.

The *rich concentrate* contains about 16 oz. gold per ton, and is treated only once a month. It is mainly arsenical pyrites. A lime-water wash is given, after which the concentrate is dried by vacuum filtration and dumps onto a floor where it is allowed to air for 24 hours preceding cyanidation. This operation is conducted in agitator tanks; working strength of solution, 0.3 per cent KCN; proportion of liquor to concentrate, 1:1; time of first treatment, 12 hours. They are then filtered, washed and retreated with cyanide for 15 hours. The first treatment reduces the value to about 1 oz. gold, and the second to about 0.5 oz.; KCN consumption, 11 lb. per ton; zinc, 2 lb. per ton. The final residue is stored for six months and again run through the mill mixed with ore.

The *ordinary concentrate* is cyanided in practically the same manner as the rich concentrate. Its value is about 1.12 oz. gold per ton and gives an extraction of about 90 per cent. Proportion of solution to concentrate, 60:100; time of agitation, 22 hours; solution, 0.22 per cent KCN; consumption of KCN, 2.75 lb. per ton, and of zinc ¼ lb.

The *sand* has a value of about 80 cents to 85 cents and is treated in 24-ft. percolators served by one distributor. The time of the cycle of treatment is as follows:

Filling	16	to 18	hours
Drying and airing	4	4	"
Circulation of solution	36	36	"
Removing liquor and washing	6	" 8	"
Sluicing out	2	" 3	"
		64 to 69	hours.

When a percolator is charged with sand, sump solution of 0.08 per cent KCN is run on, followed by slightly turbid gold liquor from the slime plant. After the latter is all on, circulation of solution is started and continued for 36 hours. Charge, 115 tons to 120 tons of sand and about 40 tons of solution. The value of the residue is about 20 cents. Consumption of cyanide, 0.4 lb. per ton, and of zinc, 0.08 lb.

The *slime* is thickened and run to dewatering vats, where it is dewatered by a set of vacuum filter frames lowered into the pulp. The dewatered cake is lifted and discharged into the extraction vat previously filled with KCN solution. Three journeys of the filter frame are required to fill one extraction vat. After agitation in this vat, the pulp is run into another tank, where the same vacuum procedure is carried out to remove the gold solution from the slime, which is washed and discharged as usual. The average value of the slime is about $2 per ton, and an extraction of 79 per cent is obtained.

Excellent precipitation is obtained in the zinc boxes, the sump solutions rarely running more than a trace of gold. Ninety per cent of the recovery is made in the first compartment. The precipitate is treated with H_2SO_4, filtered, roasted, fluxed and melted, the resulting bullion running 940 to 960 fine. The free gold recovered on the blankets and in the mortar boxes is refined and melted, making bullion over 900 fine.

Crushing and Grinding Devices.—In an endeavor to find a suitable combination of coarse and fine crushing machines, both working under the best conditions, Mr. G. A. DENNY presents in the November, 1910, issue of the *Mexican Mining Journal* some observations on the subject. After reviewing the history of the stamp, the author finds it remarkable that this device has retained its position of eminence as a grinder. He shows that the only practical development in stamp machinery is in the increase in weight, without sufficient evidence adduced to show that the increase is necessary or efficient. He criticises the manner of calculating the mechanical efficiency of a stamp, stating that no allowance is made for the work done by previous machines, whereas in calculating the mechanical

efficiency of the tube mill the custom is to deduct the mechanical energy already expended by the stamps on the tube-mill feed.

The efficiency of the stamp falls off rapidly when crushing through fine screens. The author shows this by comparing an 1150-lb. stamp crushing 9 tons per day through an 8-mesh screen, with a 900-lb. stamp crushing 4 tons per day through a 20-mesh screen. The mechanical efficiency of the former is calculated by Stadler at 70.4 per cent and of the latter (by the author) at 45 per cent, ignoring the work of previous machines in each case. The author assails the economic usefulness of the stamp at both limits of its functions, coarse and fine crushing, showing that breakers are more efficient coarse crushers and other machines more efficient fine grinders, so that there must be a critical point where the stamp performs its function most efficiently. The author's opinion is that the economic weight of stamps is from 1000 lb. to 1250 lb., and the best average mesh is 20.

Concerning Rand practice of heavy stamps and tube mills, the author finds room for criticism in that the material delivered by the stamp at 70 per cent efficiency is fed to a machine that completes the grinding at only 17 per cent efficiency. Since the heavy stamp reduces only 25 per cent of its product to the required degree of fineness, it leaves 75 per cent of the material to be retreated in a machine of lower efficiency. He believes that lighter stamps would reduce 75 per cent of the product to the proper fineness, leaving only 25 per cent to be retreated in the mill of lower efficiency.

Iron and Steel.

Composition of Some Tool Steels.—According to L. Klusmeyer, in the composition of tool steels the following elements influence the quality of the steel in a marked degree: Phosphorus lowers the resisting power of steel against impacts and makes it cold-short brittle. It gives a coarse crystalline structure to the steel. Sulphur lowers the strength of steel at a dark-red heat and makes it red-short brittle. Silicon lowers the coefficient of expansion. Manganese in small quantities eliminates the bad influence of sulphur and silicon. In too great quantities it makes the steel brittle. Chromium and tungsten increase hardness without influencing the expansion in the same measure as carbon does. Chromium and tungsten steels at same hardness are stronger than carbon steels. The percentage of phosphorus in first-class steel is at times not more than 0.008 per cent, and moves within the limits of 0.01 per cent to 0.03 per cent; sulphur percentage is 0.03 at the most and copper is 0.25. As manganese raises the malleability of steel, from 0.2 per cent-0.3 per cent is added. Silicon, in amounts of 0.2 per cent, facilitates the production of good castings. The percentage of carbon wavers between 0.75 per cent and 1.50 per cent, and depends on the amount of hardness desired. Following is a comparative table of several steels:

	Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.
Stirian tool steel (Steiermark)	1.12	not deter.	0.23	0.023	0.016
Tool steel—Etienne (Ledebur)	1.00	0.06	0.08	0.02	0.015
Tool steel—Kappenberg (Ledebur)	0.92	0.09	0.12	0.02	0.005
Sheffield steel (Bischoff)	0.75	...	0.23	0.04	0.022
German crucible steel (Bischoff)	1.24	not deter.	0.15	0.016	0.016

Manganese, chromium and tungsten increase the natural hardness of steel in a marked degree; silicon and phosphorus increase it slightly. High-speed steels have a large percentage of chromium and tungsten with a small percentage of manganese and silicon.

Following is a table of the composition of chromium-tungsten high-speed steels:

	Chromium.	Tungsten.	Carbon.	Manganese.	Silicon.
Bedel & Co., Paris	8.10	22.80	0.90	0.47	0.20
Austrian Phönix steel	3.70	20.70	0.67	0.14	0.15
English Novo steel	2.95	18.85	0.76	0.42	0.33
Austrian high-speed steel	7.19	24.50	0.93	0.23	0.24
Universal steel	6.50	23.50	0.60	0.14	0.48
Kysar steel	1.22	18.77	0.87	0.27	0.11
Armstrong steel	3.40	12.44	0.78	0.49	0.44
German steel	3.70	30.20	0.60	0.24	0.23

(*Werkstatts Technik*, October, 1910.)

Electrical Steel Furnace at Dommelsdingen.—Mr. Herman Thieme gives a description of the new electric steel furnaces of the Gallais-Metz Company, in Luxembourg, in *Elektrotechnische Zeitschrift*, Sept. 8 and 15. The electric-steel plant has two single-phase induction furnaces of three or four tons capacity and one three-phase induction furnace of 1.5 ton capacity. The tilting devices of the furnaces are controlled by a 10-hp motor. To safeguard against tilting the furnace at too great an angle an electrically operated differential band brake stops the tilting device at the proper angle of about 45 degrees, and when tilting back to draw slag at an angle of 17 degrees from the horizontal. The furnace chamber is built up of iron plate 18 mm thick, surrounding the transformer cores and lined inside with firebrick. After the firebrick lining has been dried, magnesite or dolomite, with tar as binding material, is stamped in. To dry and preheat the lining, wrought-iron rings are placed in the furnace and are heated by induced currents to a bright-red heat. About 180 kw are employed for this preliminary drying and heating operation, which lasts about eight hours; this is for the single-phase furnaces which consume 380 kw in normal operation. The furnaces are started by means of oil switches from the charging platform. All instruments and regulators and controlling devices are installed on the furnace platform. A special advantage of the three-phase furnace over the single-phase furnace is that the rotary magnetic field produced in the former provides an excellent mixing and stirring of the charge.

Zinc.

Condensation of Zinc Vapors.—In *Metallurgie* of October 8 Mr. A. Roitzheim discusses the condensation of zinc vapors to liquid zinc and the reasons for the formation of zinc dust. He finds that to obtain zinc in liquid form it is all important to cool the zinc very slowly and gradually from the higher temperature at the entrance into the condenser to the temperature of liquid zinc. If the vapor is cooled too suddenly zinc dust is obtained. The dilution of the zinc vapors by carbon monoxide in the retort does not seem to offer difficulties with respect to condensation. But the carbon monoxide seems to have a certain effect on the structure of the zinc. As to the temperature variations from muffle to condenser the following data is given: The reduction of the ore in the muffle takes place according to the equation $2\,ZnO + C_2 = 2\,Zn + 2\,CO$. The two terms on the right-hand represent the gas mixture which enters into the condenser. Under normal atmospheric pressure the partial pressure of the zinc vapor is, therefore, 388 mm mercury. At this pressure a temperature of 867° C. is required to have equilibrium between liquid zinc and zinc vapor (this temperature being obtained by graphical interpolation from figures given in R. Schenck's "Phys. Chemie der Metalle," page 5). Since the temperature of reduction of zinc oxide in the muffle is in practice between 900° C. and 1400° C. there is no danger of the formation of liquid zinc in the muffle, and the entrance to the condenser is also heated above 867°. The other end of the condenser has a temperature of 500° C. At this temperature the tension of the zinc vapor is only about 5 mm mercury, so that by far the largest part of the zinc has been condensed. The balance is still in the form of vapor and is caught outside as zinc dust. The volume of this "lost" zinc vapor is $V_{Zn} = 5/755\ V_{CO} = 1/151\ V_{CO}$. It is thus seen that theoretically 100/151, or 0.662 per cent of the zinc content of the charge, would be obtained as zinc dust. Under correct operation in practice 3 per cent are obtained in form of dust, due to the difficulty of maintaining proper conditions of operation.

Influence of Steam and Hydrocarbons on the Roasting of Zinc Blende.—Mr. Felix Thomas in *Metallurgie* of October 8 and 22 reviews the results of experiments conducted by him on the influence of steam and hydrocarbons on the roasting of zinc blende. The favorable results obtained by briquetting zinc blende with some binding material, rich in hydrocarbons,

and by the use of air saturated with steam for reducing the ore, caused investigations on the influence of steam on ZnS and its products of oxidation. The investigations led to the following results: Decomposition of $ZnSO_4$ by steam begins at 600° with moderate speed and ends at 820° quickly and can, therefore, be carried out in practice at a temperature lower by 100° than when dry air is used. Steam begins to act on Zns at 650°, and under the conditions in which the decomposition of the sulphate in a steam-saturated atmosphere is completed the decomposition of ZnS is 30 per cent when the air is shut off. This observation would make a simplification of the present roasting process possible under the following conditions: The ore briquetted with a fuel containing hydrogen, for instance, tar. The ore briquets are roasted by the oxidizing flame of a fuel rich in hydrogen (gas, by-products of oil distillation, producer-gas, water-gas, etc.) at a temperature from 700° to 750°; near the end of process, the temperature is raised to 820°.

A marked acceleration of the roasting process is gained if a good furnace is chosen, for instance, with movable grates, so that the roasted surface layers of the briquets are scraped off from the unroasted core in the center and are brought into the hottest part of furnace, where the last traces of sulphates and sulphides will be removed. The practical advantages which can be obtained are a lower roasting temperature, and, therefore, less loss of ZnO, less reduction of ZnO to Zn and less danger of sintering and smelting through agency or metallic sulphides, Cu_2S, PbS, Sb_2, S_2, etc.; greater economy of time and fuel, especially if the roasted material is removed immediately, as suggested above, the possibility of using to good advantage mechanical roasting furnaces. Manual labor, as used at present, can be eliminated entirely. At the low temperatures used in the process iron parts could be used for the furnace without danger. More satisfactory results are obtained, since the roasted material is separated from the unroasted material, and the possibility of retaining sulphur is not so great.

Bisulphide Process.—Reference has been made previously in these columns (Vol. VIII, No. 10, October, 1910, pages 591 and 595) to a process for the treatment of zinc-lead ores by roasting and leaching with sulphurous acid. In this connection the *Australian Min. and Eng. Rev.* chronicles in its issue for Sept. 5, 1910, the plans for construction of a plant at Rosebery to treat ores by this process. The details are practically the same as those indicated in our former synopsis and patent review.

Aluminium.

Use of Aluminium in the Gas Industry.—The *Revue Des Mataux Et Alliages*, of October, 1910, brings an article on the use of aluminium in the gas industry. Aluminium seems to be unaffected by sulphuric gases, sulphuric acid, hydrogen sulphides and gases generated in the gas retorts, which are more or less compounds of the cyanogen group of gases. It is not attacked by ammonia compounds either, and, therefore, well adapted to the replacing of copper in many ways. Compared to brass aluminium offers an even greater resistance to the sulphur elements in gases, and can be substituted for brass and bronze with advantage. Aluminium tubes of convenient diameter could easily be substituted for metal now used in gas fixtures and installations. Where aluminium installations have been tried they have given excellent results and have proved more economical than brass fixtures. The metal is stronger, of good appearance, makes further decorations unnecessary, and is not affected by the effects of the combustion of gas. Aluminium has been used in French gas plants for manometers and measuring instruments and has proved a great success.

Chemical Engineering.

Oil Shale Industry in Australia.—Early references to oil shale deposits in New South Wales date from 1807. In the Sept. 5, 1910, issue of the *Aust. Min. and Eng. Rev.*, occurs a description of the most recent developments of the oil shale deposits there, and the plant in which they are treated. The Commonwealth Oil Corporation has crude oil and refining works at Torbane and Hartley, respectively, and has in course of erection extensive plants at Newnes.

The richer portion of the shale is exported for gas enrichment, and the remainder is crushed and charged into retorts of which there are two benches of 32 and 40 retorts, respectively. Producer gas is used as fuel. Blowers exhaust the oil vapors from the retorts through mains and a series of condensers, where the gases are cooled and the crude oil drained off. The ammonia is removed in scrubbers and the uncondensable portion of the gas is returned to the retorts as fuel. The crude oil is shipped to Hartley from Torbane, where it is treated with sulphuric acid to remove tar, and then with caustic soda to neutralize the acid. The oil is then distilled with steam, and by fractional distillation the various products are obtained. The paraffin base is filter-pressed to separate it from contained oil, after which it is solidified into cakes, which are treated in the sweating-room until the wax is of a whitish color. It is subsequently treated with floridine prior to filtration, after which it is ready for candle making. A residue from the stills is utilized as carbonaceous fuel.

At the Newnes works the first bench of retorts is completed. It comprises 64 retorts, 56 ft. high, the lower half being of firebrick and the upper of iron. Producer gas for firing is consumed around the brick column, and the uncondensable gases from the distillation of the shale are also returned to the retorts for fuel; in fact, once the retorts are started the shale itself produces almost sufficient fuel gas. The operation of the retorts is continuous, the shale being fed at the top and discharged continuously at the bottom by means of automatic drawing gear. The paraffin extraction department is equipped with an ammonia refrigerating plant. The tar removed by treatment with acid and alkali is used as fuel in the distillation department.

Radioactivity.

A New Radioactive Mineral.—A new radioactive mineral named Pilbarite, frim its discovery and occurrence in the Pilbarra gold field, Australia, is described in the Sept. 7, 1910, issue of the *Aust. Min. Standard* by Mr. EDWARD S. SIMPSON. The mineral occurs in a tantalite lode in nodules varying in size from that of a pea to a walnut, and has a bright canary-yellow color. Other properties are: Specific gravity, 4.4 to 4.7; hardness, 2.5 to 3; composition, hydrated silicate of lead, uranium and thorium; uranium trioxide, 27.09; thoria, 31.34; lead oxide, 17.26; silica, 12.72, and water, 7.56 per cent. According to Rutherford and Boltwood's value for the ratio of radium to uranium in such minerals, Pilbarite would contain about 7.5 centigrams of radium per ton. Under treatment by acids the mineral gives off gas which is neither oxygen nor carbon dioxide, and is assumed to be helium, although the author was unable to prepare spectrum tubes of it. Pilbarite is strongly radioactive, and when placed in a Glews scintilloscope yields a rapid succession of flashes due to the impact of emanations on the zinc sulphide screen.

The Amethyst Coloration of Crystal Glass.—In some notes on radium research Mr. J. H. NIEMANN advances the idea that radium is the main agent in the amethyst coloration of glass. The article appears in full in the *Aust. Min. Standard* for Sept. 21, 1910. The coloration is usually ascribed to the slow peroxidation of manganese or to the effect of violet solar rays. The author finds, however, that this coloration will occur under conditions which preclude the acceptance of these explanations. Glass becomes colored in some sections much more rapidly than in others; the colored glass acts radioactively, stimulating the growth of plants, and darkening photographic plates; the colored glass decomposes water, and commonly explodes after having been highly colored. The author regards these phenomena as tending toward his idea that radium emanations are responsible for the coloration of the glass.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute.

At the autumn meeting of this institute at Buxton the reading and discussion of papers extended over three days, and the chair was occupied by the president, the Duke of Devonshire.

In opening the discussion on Mr. C. de Schwarz's paper, entitled "*The Briquetting of Iron Ores*," Prof. Henry Louis dissented from the author's view that the use of more effective explosives increased the proportion of small and dust ore, and said that the author appeared to be under a misapprehension as to the nature of the Gröndal invention, which was in use in Great Britain, Spain, Norway, the United States and, he believed, also in Germany, as well as in Sweden. When nodules were properly made in the rotating furnace they equaled the best briquets. With reference to the process in which the ores are mixed with spathic ore and lime for briquetting, his investigations did not confirm the author's statement as to the reaction which took place.

Mr. S. G. Robinson thought that the process described had been copied from the Portland cement rotary kilns, the great merit of which was cheapness of operation.

Mr. J. E. Stead said briquets were not made firm enough to stand transport without the outsides breaking up and producing 15 per cent to 20 per cent of dust. Organic compounds were suitable with magnetic oxides, but not with peroxides, the oxygen from which consumed the binding material very soon after the briquets were put into the furnace.

Mr. G. E. Moore having read his paper on "*The Manufacture of Rolled H-Beams*," Mr. Andrew Lamberton said rolling-mill owners in Britain had not ignored the subject, but rolled H-beams were not produced here in large quantities, and there was not any mill of the type which the author had described, because it was questionable whether there was a market in this country for beams much larger than those which were commercially produced at the present. The built-up girder was coming into favor with engineers who considered the commercial as well as the professional aspect of the matter.

Mr. Moore, in reply, said H-beams up to 15 in. x 15 in. were imported into England, and he thought that they might be manufactured here.

Mr. E. Colvert-Glauert read a paper by Dr. S. Hilpert, of Berlin, and himself, on "*Sulphurous Acid as a Metallographic Etching Medium*."

Prof. A. McWilliam remarked that Mr. Colvert-Glauert was a Sheffield student, and he was interested in the fact that one of his photo-micrographs corroborated some of the views expressed in a paper by Prof. Arnold and himself, treating on the thermal transformations of carbon steels, which had not been accorded general acceptance by other metallurgists.

Mr. E. H. Saniter said his experiments with sulphurous acid had not been successful in consequence of the great staining effect it had on the specimens.

Mr. W. H. Hatfield asked for a definition of austenite and hardenite, which the authors mentioned as occurring in the same sections.

Mr. Colvert-Glauert replied that sulphurous acid used as described (a 3 per cent to 4 per cent solution of a saturated solution of SO_2 in water) did not produce staining unless the specimen were unduly exposed to the air before being rinsed. As to austenite and hardenite in the same section, he did not regard austenite as a true constituent.

Dr. S. Hilpert's paper on "*The Preparation of Magnetic Oxides of Iron from Aqueous Solutions*" evoked no discussion. It was of a purely chemical nature, and the author's main conclusions were that the production of magnetic oxide of iron Fe_3O_4 from aqueous solutions is only possible through the precipitated FeO dissolving in the ammonium residue; that the magnetic properties of Fe_3O_4 are derived from the acid property of Fe_2O_3, and that the true magnetic oxide of iron is Fe_2O_4 in the form of feric ferrate.

On the second day the proceedings opened with the reading of a paper by Mr. Arthur Hague and Prof. Turner on "*The Influence of Silicon on Pure Cast Iron*," which was followed by Mr. H. I. Coe's paper on "*Manganese in Cast Iron and the Volume Changes During Casting*," and the two papers were discussed together.

Mr. E. H. Saniter said that it was an interesting fact, which was brought out in both papers, that manganese increased the brittleness of cast iron. It was his experience that if ingot molds contained more than 0.5 of manganese they cracked much sooner than those with a lower percentage of manganese.

Mr. J. E. Stead was of opinion that the research conducted by the authors had not gone far enough to fully elucidate the facts. It was noteworthy that small variations in the amount of manganese caused such differences in the expansion; but, as Prof. Turner had stated, phosphorus, sulphur and other impurities behaved similarly.

Mr. E. Adamson said that with regard to hardness tests the combined carbon was generally taken as an index of hardness; but the statement that the softest bar was obtained with 0.97 and 1.04 of combined carbon was not in agreement with general experience; nor was the statement that hardness was increased after 2 per cent of silicon.

Mr. W. H. Hatfield remarked that he had demonstrated that the addition of silicon had no effect on the carbon until a certain percentage had been added, when the change occurred quite suddenly.

Mr. F. W. Harbord was of opinion that the hardening effect of silicon up to 1 per cent only applied to hard steels and was due to the carbon. On steel free from carbon, silicon would not have the same hardening effect.

Prof. Turner, in his reply, agreed with Mr. Stead's remarks as to the necessity of finding the true explanation of the accumulated facts, but he thought further research was necessary. The view advanced by Mr. Adamson that silicon did not cause a reduction of the total carbon was at all events only right within the limit of 4 per cent.

The discussion on Mr. C. A. Edward's paper on "*The Theory of Hardening Carbon Steels*" was opened by Prof. Arnold, who said he believed the propositions put forward would meet with a general acceptance and would tend to reconcile the differences between the carbonist and the allotropist schools. Still it was necessary that theories should be based upon scientifically proved facts, and it was essential that before attempting to formulate a definite theory an agreement should be come to as to the method of taking the cooling curve of pure iron, and he trusted that this would soon be done. A curve taken this year at Charlottenburg with very delicate apparatus had given for pure iron practically a reproduction of a curve obtained at Sheffield 14 years ago.

Mr. E. H. Saniter, with reference to the suggested practical identity of austenite and martensite, asked if it were not the fact that in carbon steels austenite was much softer than martensite?

Prof. Turner was inclined to question Mr. Edwards' statements on the subject of twin crystallization, because evidence was lacking. Twin crystals were ordinarily produced only as the consequence of annealing, and he could not concede that pressure alone was adequate.

Dr. Carpenter suggested that the details for the method of taking the cooling curves of iron should be referred to some scientific body such as the Physical Society.

Mr. J. E. Stead said M. Osmond was the discoverer of what he called austenite, and he thought that his opinion should be asked with regard to twinning. He did not find anything of the sort in ordinary carbon steel; and concerning the pressure theory he asked if Mr. Edwards had realized that heating a bar of steel to redness and quenching it produced tension on

the outside, and not pressure? If Mr. Edwards' theory were true such steel should be soft on the outside and hard inside.

Prof. A. McWilliam thought that metallurgists would not accept Dr. Carpenter's suggestion to leave the decision as to the method of taking cooling curves to the Physical Society or any other society consisting of amateurs in such matters. Personally he would not accept a decision on the point from a special committee of the Iron and Steel Institute, but would take the curves in the way he found best.

The discussion was adjourned to the following day when Mr. Edwards, in the course of his reply, said he had no definite conviction as to the best method of taking the cooling curves. In view of the fact that austenite was regarded as softer than martensite it was noteworthy that while experimenting on a hardened, high-speed steel, which would be supposed to consist of pure austenite, he had obtained a hardness figure of 866 with the Brinell test, which he thought was superior to any ever yielded by any quenched carbon steel.

Mr. D. F. CAMPBELL's paper on "*Electric Steel Refining*" was then taken.

Mr. E. H. Saniter opened the discussion and expressed his opinion that, while there was a great future for the electric furnace, he did not think it was likely to come into general use for the production of steel rails; and his opinion was emphasized by the fact that at South Chicago, where the furnace had been working some 15 months on rail steel, it might have been expected that if the production had been economical more furnaces would have been put into operation to deal with the entire output of rails at those works.

Mr. L. Greiner congratulated Mr. Campbell on not having advocated any particular type of furnace.

Mr. J. H. Heap said the question was purely commercial, and he thought there were intermediate stages at which the use of the electric furnace would be advantageous. Obviously it could be applied to the refining of common Bessemer steel. It was desirable to have analyses of the finished steels produced in the last three months by the Illinois Steel Company, because statements had recently been made that those steels were only of average good quality. He suggested that the question of electric refining should be investigated by a committee of the Institute.

Mr. F. W. Harbord considered that it was beyond question that the electric furnace could compete commercially with the crucible furnace very easily. The main question was the quality of the product, and he thought it would be conceded that for 90 per cent of the purposes to which tool steel was applied the electrically produced steel fulfilled all requirements. The electric furnace was dear for melting, but refined economically. For high-priced material it could only be employed commercially when low sulphur and phosphorous were essential.

The Corrosion of Wrought Iron and of Steel.

Mr. WALTER MACFARLANE in his recent presidential address to the Staffordshire Iron and Steel Institute said engineers were now becoming impressed with the capacity of wrought iron for resisting corrosion. In breaking up old iron ships the condition of the metal contrasted very favorably with that of steel from ships which had had a less period of service than the iron vessels. During the overhauling of a railway bridge several years ago, over the Clyde, the condition of the iron members was found much better than that of the steel members. In the west of Scotland the output of wrought iron was greater to-day than ever it was before; and there was difficulty in getting enough puddlers. While it could not be expected that the output of wrought iron would keep pace with the enormous amount of mild steel for large structural work it was probable that the use of iron for special purposes would become more extensive.

High-Pressure Gas in the Casting of Metals.

The Birmingham Aluminium Casting Company has experimented with high-pressure gas for melting aluminium with satisfactory results—a charge of 100 lb. having been melted in slightly more than one hour in a furnace especially constructed for the purpose. Only aluminium has been dealt with at present, but it is proposed to extend the experiments to other metals and alloys.

Structural Steel for Ships.

The Engineering Standards Committee has issued a revised specification for structural steel for shipbuilding. The maximum tensile strength for plates for cold flanging is reduced from 32 tons to 30 tons per square inch. The minimum tensile strength and the elongation remain unaltered. For bars more than 1 in. in diameter, whether tested full size as from the rolls or after having been turned down, the employment of the four-diameter test piece is provided for.

Market Prices.

October, 1910.

Copper opened at £55.10, rose fairly steadily to 18th, when at £57, then fell off slightly, closing at £56. 13. 6.

Tin opened at £156, rose sharply to the 5th, when £162, then, with two small drops, rapidly to £170, on the 20th; thence declining as rapidly to £162 on the 25th; finally closing at £165. 15. 0.

Lead £13. 2. 6. at opening, rising steadily throughout the month to £13. 12. 6.

Haematite had fallen to 64/6 by the 1st and touched 63/9 on the 6th. Thence it rose and hardened, remaining at 65/- to 65/3 until the 25th, closing at 65/-.

Scotch Pig opened at 54/-, but rose again to 54/10½ by the 10th and to 55/7½ on the 12th, remaining in the neighborhood of this price to the 25th, and closing (26th) at 55/9.

Cleveland Warrants were down to 48/- on the 1st, recovered to 48/10 by the 10th, and have kept near that figure, closing (26th) at 49/11.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder	24	0	0
Borax, British refined crystal	16	0	0
Sulphate of ammonia, f. o. b. Liverpool, per ton	13	0	0
Sal ammoniac, lump, firsts, delivered U. K. per ton	42	0	0
Copper sulphate, per ton	19	10	0
Camphor, 1-oz. tablets		1	7
Caustic soda, ash, 48 per cent ordinary, per ton	5	10	0
Shellac, standard, T. N. orange spots, per cwt	4	2	6
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid, 97/99 per cent, per gal			11½
Hydrochloric acid, per cwt		5	0
Creosote, ordinary good liquid, per gal			2⅛
Naphtha, solvent, 90 per cent at 160° C., per gal			11
Petroleum, Russian, spot, per gal			5¾
Copper ore, 10 per cent to 25 per cent unit	9/1½ to 10/1½		
Tin ore, 70 per cent, per ton	£104 to £106		
Mica, small slab, in original cases, per lb	6d. to 1/-		

	£	s.	d.
Litharge, flake, per ton	15	5	0
Quicksilver, per bottle	8	7	6
Zinc sheets, V. M., f. o. b. Antwerp	29	7	6
Red lead, genuine, per ton	15	15	0
Bleaching powder, 35 per cent, per ton	4	5	0
Platinum, per oz. nominal	7	14	0

India Rubber, Para fine, is still somewhat unsteady, though more about a natural level. By the 5th it had fallen to 5/11, and risen again by mid-month to 6/3, dropping to 5/10 by the 26th, and still at this price.

Sulphate of ammonia has risen 5/-.

Copper sulphate is up 30/-.

Bleaching powder is brisk at Newcastle, being inquired for at £4. 15. 0.

Shellac has risen 1/6.

Platinum is still rising in the quotation. Tin ore has risen £9. Litharge is 5/- dearer. Red lead has risen 5/-.

Canal Zone Meeting of the American Institute of Mining Engineers.

Eighty members of the Institute, with their wives and guests, making up a party of 120, have completed a highly successful trip to Panama. The passenger accommodations of the Hamburg-American Atlas Line steamer *Prinz August Wilhelm* were reserved for the sole use of the party, going and returning, and with the Institute's cruising flag flying from the masthead an auspicious start was made from New York on Oct. 21. The general plan of the meeting was to hold sessions of the Institute for reading of papers and discussions on the steamer and at the Hotel Tivoli at Panama, the other incidentals being a week spent in inspecting the canal operations, 36 hours ashore at Kingston, Jamaica, and 24 hours at Havana.

At Havana members of the Institute and their friends received the visitors with enthusiasm, showing them the sights of the city and arranging a presentation to President Gomez.

PHOTOGRAPH OF MEMBERS OF AMERICAN INSTITUTE OF MINING ENGINEERS ON THEIR PANAMA TRIP.

At Kingston two very interesting excursions were arranged, one to the unique botanical garden at Castleton and another to the old capital, Spanish Town, and the gorge of the Rio Cobre, with a final dinner at the magnificently situated Hotel Myrtle Bank on the invitation of President Brunton, of the Institute.

But it was on the Isthmus that the interest and enjoyment reached their culmination. Colonel Goethals and his staff received the visitors with the utmost cordiality, placing themselves at their constant service during the whole week's stay. A special train conveyed the party each day to the part of the canal to be inspected, and with the "bosses of the job" to give intelligent and satisfactory answers to every question asked, the opportunities for obtaining thorough and accurate information upon every possible phase of the work were absolutely complete. The admiration expressed by the visiting engineers upon the conduct of the canal work was open and sincere, and found spontaneous and concrete expression on the homeward trip in the following resolutions, which, signed by every one of the engineers in the party, cannot but be regarded as a remarkable endorsement of the wisdom of the plan of the canal and of the wonderful energy with which the work is being prosecuted:

"We, the undersigned, members and guests of the American Institute of Mining Engineers, after a visit to the Isthmus of Panama, and inspection of the work of the United States Isthmian Canal Commission, and after full discussion of our individual impressions, find ourselves in unanimous agreement as to the following conclusions:

"The present plan of the work is clearly practicable and the best, in our judgment, that could be devised under the conditions imposed. It is perhaps a question whether by the choice of a higher level some of the difficulties and uncertainties of excavation in the Culebra Cut might not have been minimized, but a higher level has its disadvantages also, and no one now seriously proposes such a plan. On the other hand, we are convinced that a canal at a lower level, and especially at sea level, is practically out of the question; that no man can estimate its cost, or even guarantee its satisfactory completion and maintenance at any cost. We are satisfied that the sea-level canal, as proposed, if actually completed, would be inferior to the present lock canal by reason of its necessarily narrow and tortuous channel, its liability to many disturbances from which the lock canal is comparatively free, and so forth. The experience gained in the Culebra Cut throws additional light upon the sea-level plan and renders that scheme less worthy of approval by engineers than it was when with less information some eminent authorities favored it. In a word, we do not think that any prudent engineer would now recommend the deepening of the Culebra Cut below the level now fixed for it.

"The creation of the great Gatun Lake by means of the Gatun Dam seems to us to be the best possible way of dealing with the floods of the Chagres and other streams. The location of the Gatun Dam, spillway and locks is singularly favorable for such constructions, and there is, in our judgment, no reason for any anxiety as to their stability.

"The one serious remaining problem is presented by the nature of the ground in the Culebra Cut. There have been extensive slides on the sides of this excavation, and more of them

are to be expected, but they involve nothing more than the cost and delay of removing the material which they will force into the cut. They will ultimately end, and we regard as reasonable the calculation of the engineers in charge as to the time and money which they may call for. The results of these calculations are included in the estimates of the commission as to the cost of the canal and the date of its completion.

"We are unanimous in our praise of the manner in which sanitation, transportation and construction are performed with rapidity, skill and economy. A spirit of loyalty, emulation, industry and pride seems to animate employees and officers alike. This spirit, so difficult to arouse among workers in tropical climates, is due in this case to two causes: first, the inspiring example of Colonel Goethals and his associates, and, secondly, the splendid work of the sanitation department under Colonel Gorgas. The cities of Panama and Colon, though politically outside of the Canal Zone, have shared in the benefits of the sanitary administration and reflect an unwonted cleanliness, comfort and safety.

"We acknowledge the entire freedom and fullness with which everything we desired to see was shown to us, and everything we desired to know was told to us by the officers of the commission. There was evidently no wish to withhold or conceal anything. On the contrary, enquiry and criticism were frankly sought and heartily welcomed.

"This is but a meager summary of the points on which we are agreed. The details of individual opinion will appear later in the published report of our discussions. Meanwhile, we unite in this common declaration, which covers our conclusions on all main points. We think the present plan of the canal is good; that the work is in thoroughly capable hands; that it is progressing satisfactorily, and that it will be completed by the date set for it, Jan. 1, 1915, and probably earlier, provided Colonel Goethels and his associates receive the hearty support of the American people and its representatives in Congress. The canal engineers are the right men in the right place. The great work in which they are engaged is not connected with partisan politics, and citizens of all parties should combine to secure its early and triumphant completion. In that consummation every American should take greater pride than in any victory of military or political conflict."

[Signed by the 80 engineers (mining, civil, electrical, metallurgical and mechanical) in the visiting party, all endorsing the above and not one dissenting.]

The social features of the stay on the Isthmus included a reception by President Arosemena, of the Panama Republic, and a brilliant ball at the Hotel Tivoli, given by the visitors in honor of the officials in charge of the canal work, and which was also attended by the local government representatives and consuls.

On the return trip the U. S. S. *Tennessee* conveying President Taft to the Isthmus was met and the following message sent to the president:

"American Mining Engineers returning from Isthmus congratulate you upon good plan, splendid management and satisfactory condition of work."

To which the President replied as follows:

"Your message received. It is most satisfactory to have the assurance of men who are experts that the great canal construction has been well planned and is being carried on to a successful completion. I thank you and your associates for your visit and the courtesy of your kind message.

"WILLIAM H. TAFT."

We may without hesitation conclude that not only was the meeting one of the most interesting in the history of the Institute, but that it was of the highest public service in giving to the world an unbiased and competent opinion upon the status of the Panama Canal work such as could have been obtained in no other way.

The scientific sessions of the Institute were not particularly brilliant, although interesting to those participating. Abstracts of the papers submitted will be given in our Synopsis.

Chemists' Building in New York City.

The magnificent new Chemists' Building at 50-54 East 41st Street, New York City, is now nearing completion. It is a monument representative of the united democracy of American chemists. Without outside help it has been erected by the Chemists' Building Company, organized for this purpose by individual American chemists and chemical manufacturers.

While thus representing the ideals of American chemistry, it is a thoroughly practical twentieth-century structure with an excellent location and with excellent internal arrangements and equipment.

Fig. 1 gives an outside view of the Chemists' Building. It is a ten-story fireproof structure, gracefully designed in French

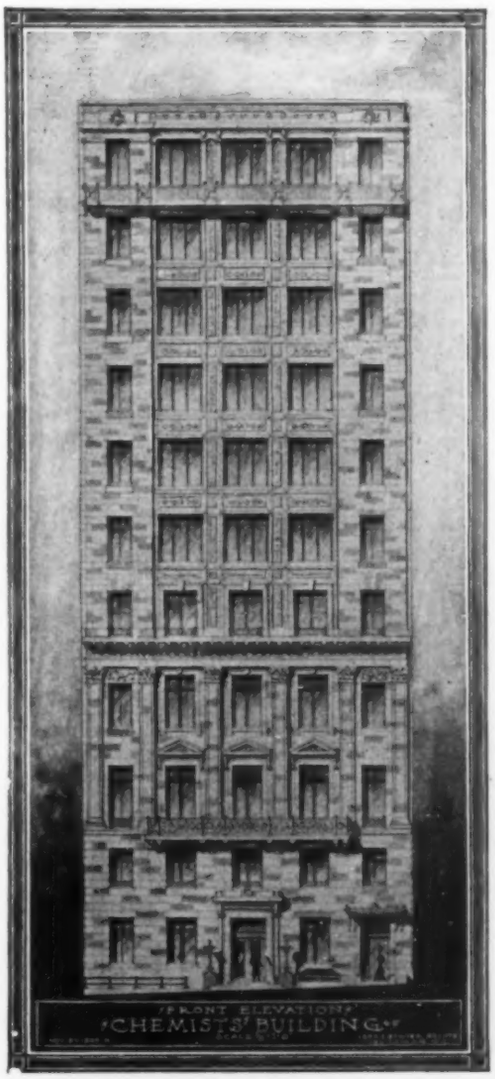


FIG. 1.—CHEMISTS' BUILDING IN NEW YORK.

Renaissance style, of the Louis XVI period. Being located in the block between Madison and Park Avenues, in 41st Street, it is close to the Grand Central Station and to the 42d Street express station of the subway, as well as to the prospective terminals of the McAdoo and Steinway tunnel systems. It is a few minutes' walk from the New York Public Library in Fifth Avenue (opposite 41st Street), from the Engineering Societies Building in 39th Street, from the Engineers' Club in 40th Street, and from the Academy of Medicine in 43d Street.

The lower five floors of the Chemists' Building are leased to the Chemists' Club and contain all the appurtenances of a social club. The first floor, a plan of which is given in Fig. 2,

contains in the rear a large auditorium with a balcony. The next floor is fitted as a lunchroom, with social rooms opening off from it. The fourth and fifth floors are to be devoted to

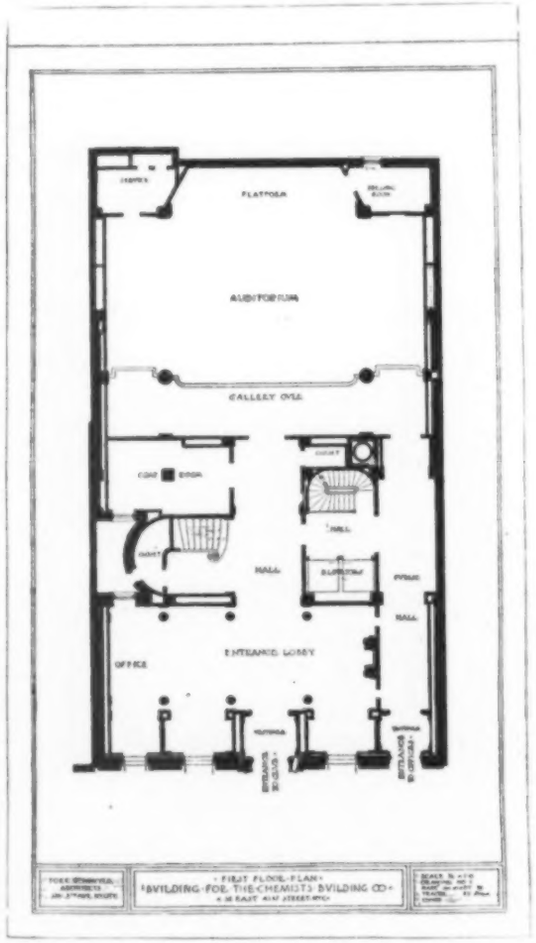


FIG. 2.—PLAN OF FIRST FLOOR.

living and sleeping rooms for the members, below which will be the library and museum.

The five upper stories from the sixth to the tenth have been especially constructed for laboratory purposes and can be rented either as entire floors or in suitable subdivisions, to analytical, commercial or research chemists, physicists, electrochemists, bacteriologists, etc., but not as manufacturing laboratories. A typical upper-floor plan with suggested subdivisions, which may, of course, be differently arranged, is shown in Fig. 3.

They are provided with ventilating flues, water, gas and electric mains, steam and compressed air lines, in suitable locations, but the tenants are expected to make their own connections and provide their own fixtures.

Rental will include janitor and elevator service, heat and electricity for lighting purposes, and the company will construct proper partition walls for the subdivision of the laboratories, according to the tenant's wishes. It is intended to charge low rentals and in many ways facilitate the prosecution of scientific and industrial research.

Among the special advantages of the internal equipment of the building are excellent light and ventilation; construction equal to that of university laboratories; command of a complete chemical library; opportunities for social and scientific intercourse; the fact that the building will be the natural meeting place of out-of-town chemists and manufacturers, for whom the Club provides excellent apartments; and finally the possibility of reduction of laboratory expenses through co-operative installation of expensive apparatus.

The value of this neighborhood for such purposes has been further enhanced by the erection of a large Physicians' and Dentists' Building alongside and the attention of physicians and pathologists is called to the advantages which the Chemists' Building laboratories could afford them for their own researches.

Until Feb. 1 communications should be addressed to Dr.

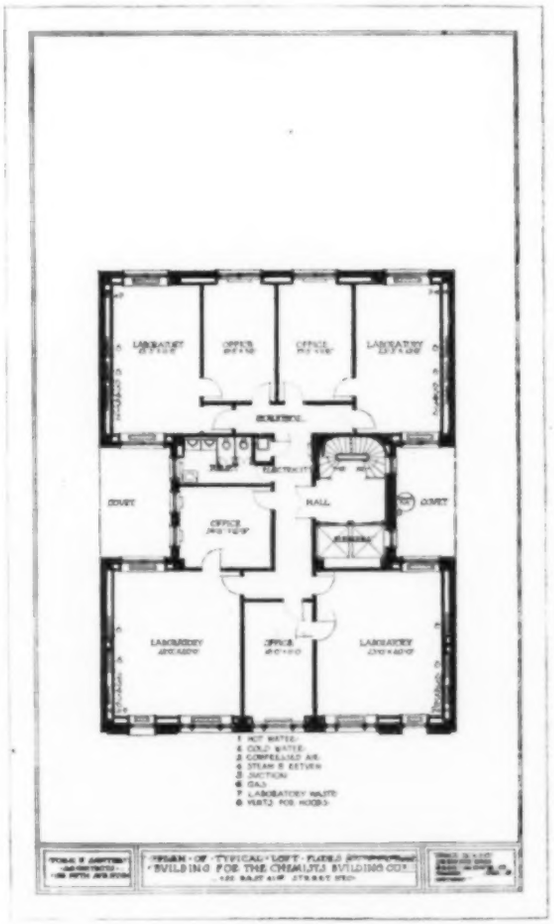


FIG. 3.—PLAN OF TYPICAL UPPER FLOORS.

Morris Loeb, President of the Chemists' Building Company, 106 West 55th Street, New York City. After that date his office will be in the building, 50 East 41st Street.

Cement Concrete Vats and Tanks.

By Albert Moyer.

Impervious, odorless, tasteless and sanitary vats and tanks for buttermilk, wine, oil, pickles, sauerkraut, etc., can be constructed of reinforced concrete, the reinforcing to be designed by a competent engineer, provided the interior surfaces are treated as follows:

After the forms are removed, grind off with a carborundum stone any projections due to the concrete seeping through the joints between the boards. Keep the surface damp for two weeks from the placing of the concrete. Wash the surface thoroughly and allow to dry. Mix up a solution of 1 part water-glass (sodium silicate) 40° Baumè, with 4 to 6 parts water, total 5 to 7 parts, according to the density of the concrete surface treated. The denser the surface the weaker should be the solution.

Apply the water-glass solution with a brush. After four hours and within 24 hours wash off the surface with clear water. Again allow the surface to dry. When dry apply another coat of the water-glass solution. After four hours and

within 24 hours again wash off the surface with clear water and allow to dry. Repeat this process for three or four coats, which should be sufficient to close up all the pores.

The water-glass (sodium silicate) which has penetrated the pores has come in contact with the alkalies in the cement and concrete and formed into an insoluble hard material, causing the surface to become very hard to a depth of ⅛ in. to ½ in. according to the density of the concrete. The excess sodium silicate which has remained on the surface not having come in contact with the alkalies is soluble, therefore easily washed off with water. The reason for washing off the surface between each coat and allowing the surface to dry is to obtain a more thorough penetration of the sodium silicate.

It is obvious that concrete surfaces so treated, if hard, impervious and insoluble, have been made impervious, tasteless, odorless and sanitary.

A New Radiation Pyrometer.

A new type of radiation pyrometer embracing several novel features has recently been devised by Mr. Richard P. Brown, of Philadelphia. As is well known, the radiation pyrometer is especially useful for measuring high temperatures, beyond the range of the thermoelectric pyrometer.

It is true, in the radiation pyrometer a sensitive thermocouple is also used, but this is not placed at the hot place the temperature of which is to be measured. The temperature is measured from a distance by means of a telescope which holds at its rear end the sensitive thermocouple. A concave mirror focuses the heat rays entering the tube of the telescope on the thermocouple, which is connected by wire to a millivoltmeter graduated in temperature degrees. The radiation pyrometer, therefore, has no part directly subjected to the high heat and there is in consequence no part to be destroyed by the furnace gases or by the high temperature. The radiation pyrometer has also the advantage of the readings being instantaneous, the slightest changes in temperature being shown immediately.

Radiation pyrometers heretofore manufactured have been of two types, the adjustable-focus and the fixed-focus instruments. In the radiation pyrometer with fixed focus to secure a correct reading of the temperature, the pyrometer tube or telescope should not be more than 30 in. distant when the furnace opening (from which the heat rays come to the telescope) is 3 in. in diameter, or 60 in. distant when the door of the furnace is 6 in. in diameter; in other words, the pyrometer telescope or tube should not be distant more than 10 times the diameter of the furnace opening, or piece of steel, the temperature of which is being measured. As long as the pyrometer tube is within these distances the temperature can be read correctly, but to determine this the distance has to be measured.

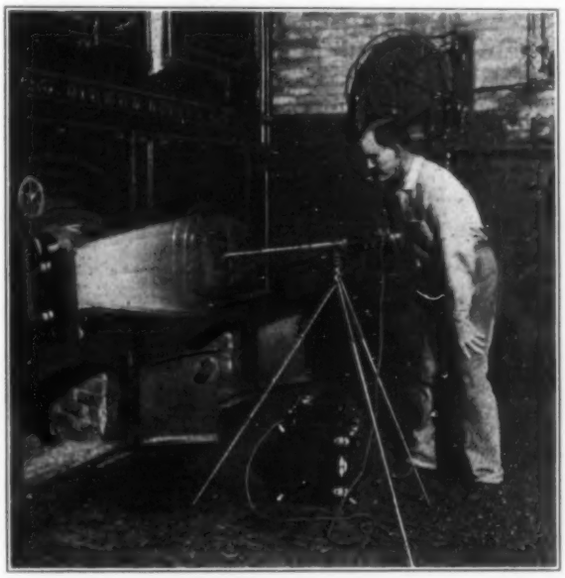
This is unnecessary with the new Brown radiation pyrometer with fixed focus, since a finder somewhat similar to those used on kodaks is placed on the tube. By means of this the tube can be readily pointed directly at the furnace opening, and at the same time the finder acts as a measure of distance, as it is only necessary that the bright red of the furnace opening takes up the whole view in the finder. If some of the dark outside wall of the furnace is showing around the bright-red opening, the tube is too far distant, and should be moved closer until the bright red only is showing. It is impossible for errors to occur with this new feature, and the instrument is stated to be accurate under ordinary working conditions within 1 per cent or 30° at 3000° Fahr.

The instrument is calibrated for measuring the temperature of a "black body," or the temperature of the walls or parts inside of furnaces which are practically "black bodies" in the thermodynamical sense. In using the pyrometer for measuring the temperature of molten metals or highly polished surfaces which reflect, a correction is made.

Another new and particularly desirable feature of the Brown radiation pyrometer is in the collapsible tube made like a telescope, which permits of the pyrometer tube, indicator, tripod and wiring being fitted in a small leather carrying case weighing complete only 15 lb. This feature is particularly desirable to engineers and superintendents who may have to take the pyrometer from one point to another occasionally for tests.

In using a radiation pyrometer focussed on a brick kiln or furnace it is frequently inconvenient, if tests of long duration are to be made, to leave the door or opening open. A firebrick tube is, therefore, inserted in the opening and projects into the kiln or furnace, the inside end of the tube being closed. The pyrometer telescope is focussed on the inner end of the firebrick tube and for permanent installation a bracket is used instead of the tripod to bolt on the wall and hold the tube.

The instrument will be adaptable particularly for measuring the temperatures about a blast-furnace plant in the open-hearth furnaces of steel works where high temperatures are employed,



RADIATION PYROMETER.

in brick kilns for burning firebrick and very refractory materials, in rotary cement kilns where the temperature is running about 3000° and must be measured 20 ft. inside the furnace, and by engineers in testing the temperature of boiler furnaces and in research work.

This new radiation pyrometer is being placed on the market by the Brown Instrument Company, of Philadelphia, with offices in Pittsburgh and Chicago, and patents have been applied for covering the advantageous features embraced in this instrument.

Lignite as Fuel.

"North Dakota Lignite as a Fuel for Power-Plant Boilers" is the title of Bulletin No. 2, just issued by the Bureau of Mines. This bulletin describes a series of tests at the pumping plant of the United States Reclamation Service at Williston, N. D. The Reclamation Service has a large project there and had installed steam boilers with furnaces designed to burn a "brown lignite" that was mined on adjacent government land.

The furnace is of the semi-gas-producer type and has an external resemblance to the so-called Dutch oven. The most striking features in its construction are the deep-set grate and the construction of the space between the bridge wall and the end of the prolonged fire brick arch. The furnace is designed to work on the gas-producer principle. The solid fuel is gasified on the grate and the gas passes through the space under the arch into the combustion chamber where most of the gaseous combustible burns.

The results of the tests on the lignite show that this fuel, though generally considered unsatisfactory, may be used with fair economy under boilers that generate their full rated capacity. In fact, when the number of heat units available is considered, the results compare favorably with those of better grades of fuel.

The tests are deemed important because the lignite deposits of the Northwest are so extensive and the distance of the region from other coal fields is so great that a large portion of the United States, including parts of North Dakota, South Dakota and Montana, may be greatly benefited by any improvements in the methods of utilizing this local fuel supply. The lignite in this field is low in heating value, some of it containing nearly 45 per cent of its weight in moisture, and it is difficult to burn in the furnaces commonly used for the better grades of coal, but the tests have shown the possibility of designing suitable furnaces for burning it profitably.

The tests were conducted by the Technologic Branch of the Geological Survey, which is now a part of the Bureau of Mines. The authors of the bulletin are D. T. Randall and Henry Kreisinger. The bulletin will be of interest to fuel engineers, especially to those located in lignite territory. It may be obtained by addressing the Director of the Bureau of Mines, Washington, D. C.

A Modern Air-Moistening Apparatus.

The moistening of air for ventilation and the removal of dust from the atmosphere is the object of the Koerting moist ventilators, an outside view of which is given in Fig. 1. The motive power in this apparatus is a fine water spray produced by forcing water (for instance, from the city water mains) through a spray nozzle (Fig. 2). The air is sucked in by this water spray and is thereby mixed with the atomized water and saturated with moisture and then forced into the discharge pipe. By this means the air in a room may be continuously passed through the apparatus and returned to the room uniformly moistened. No drops occur.

FIG. 1.—MOIST VENTILATION.

As this apparatus has no moving parts it is not subject to wear and tear. It is started by simply turning on the water. The pressure water can be taken from the city water works or can be provided by a special pump. Using a special pump the same water may be used over and over again, only the part atomized and passed into the air has to be replaced. In Fig. 1 the water enters at *C* and leaves at *D*, the air enters at *A* and leaves at *B*.

These air moisteners are used to advantage in various industrial establishments. In cutlery, cut-glass and other works where grinding, sharpening and cleaning are part of the process, the dust thrown off from the wheels is immediately drawn away by the application of these moisteners. There is the further advantage that any particles of metal or dust are precipitated as fine mud and carried away with the waste water from the exhauster and the gases are absorbed.

The moist-ventilator has also proved successful for humidifying the cheese cellars of dairies and the storerooms of dextrine or paper factories, the storerooms of cigar and tobacco factories and green houses. The regulation of the moisture is perfect, a hygrometer in the room always indicates the degree of numidity and the adjustment of the apparatus can be made accordingly. These ventilators are built with capacities from 1000 cu. ft. to 20,000 cu. ft. of air per hour by the Schutte & Koerting Company, of Philadelphia, Pa.

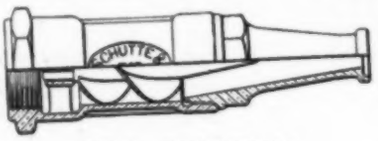


FIG. 2.—SPRAY NOZZLE.

A New Caustic Potash and Chlorine Plant at Niagara Falls.

The new plant of the Niagara Alkali Company which is to take the place of the old Roberts Chemical Company's plant is nearly completed and is expected to be in operation by Jan. 1.

The Billiter diaphragm cell, as developed under the auspices of the Siemens & Halske Company at Aschersleben, Germany, and Bruehl, Austria, will be employed for the electrolysis of potassium chloride solutions for the production of caustic potash and chlorine. The cell is "of a simple horizontal diaphragm construction, operating by percolation, thus doing away with the necessity of having a circulating brine system." It is stated that the cathode liquor of caustic potash is drawn off at a fairly high concentration, about 18 per cent, and the amp-hour efficiency in commercial operation is claimed to be about 97 per cent. It is stated that with a cell operating at 3000 amp and about 3¼ volts the production per 24 hours is 330 lb. of solid 90 per cent KOH per cell.

From the anodic chlorine gas together with the hydrogen evolved at the cathode, the Niagara Alkali Company will manufacture muriatic acid under the old Roberts patent, producing muriatic acid free from sulphur and arsenic. Part of the chlorine gas is liquified; the liquid chlorine branch of the business is handled by the Electro Bleaching Gas Company, of 24 East Twenty-first Street, New York City. It is also the intention to manufacture tin chloride, carbon tetrachloride, potassium chlorate and bleaching powder. Another branch of the business will be the manufacture of acetylene tetrachloride, chlorides and oxychlorides of metalloids and zinc chloride, and various other chlorine products.

Mr. H. D. Ruhm is president and general manager of the Niagara Alkali Company.

NOTES

Electrolysis of Water.—Hydrogen gas of practically 100 per cent purity and oxygen gas of 99.5 per cent to 99.9 per cent purity are produced from water by electrolysis by means of the Garnti process at the works of the American Oxhydrate Company, in Milwaukee, Wis. This is the only American company pushing an oxy-hydrogen welding process, while the oxy-acetylene process has found quite extended application in this country. The oxygen and hydrogen are shipped under high pressure in portable, solid-drawn, seamless, steel cylinders of 200 cu. ft. to 250 cu. ft. capacity.

The Arizona Copper Company produced during September, 1910, 1336 tons of copper.

Elmore Process.—The plant at the Sulitelma mines, Norway, produced during the month of August, 1910, 645 tons of concentrates carrying 7.38 per cent copper.

The Ray Consolidated Copper Company has issued, under date of Sept. 27, 1910, its second annual report. The company is one of those preparing to operate large deposits of low-grade copper ores in Arizona, and is building a concentrator and smelter to treat its own output. The former will be in operation early in 1911, and the latter a few months later. Coarse

crushing of the ore will be done at the mine, and ore bins will be provided at the mill to insure against interruptions. The mill will be in 8 units of 1000 tons capacity each, and will be modeled along the same lines as the large mills in Utah and Nevada. The fine grinding machinery will comprise rolls and chilian mills; the concentrating machinery of roughing and cleaning tables for the coarser sizes, and corrugated and smooth belt vanners for the finer. The entire plant will contain 16 sets of 42-in. rolls, 24 6-ft. chilian mills, 96 Garfield roughing tables, 48 Wilfley tables, and 768 6-ft. vanners, together with the necessary equipment of screening, classifying and settling apparatus. With the plant running at full capacity on a normal grade of ore the production will be not less than 80,000,000 lb. of copper per year, at a cost of 9 cents per pound.

Southern Water Power Plant.—Contract has been awarded to Messrs. J. G. White & Co., of New York, for the engineering and construction of their hydro-electric plant for the Eastern Tennessee Power Company. The dam and hydro-electric power station will be constructed on the Goose River, at Parksville, Polk County, Tenn., about 12 miles from Cleveland, Tenn. The power equipment will consist of four main units of normal rated capacity of 3000 kw each, of 200-kw exciter units and the necessary step-up transformers, etc. Each main unit will consist of one tandem horizontal turbine in enclosed case direct connected to a three-phase, 60-cycle, water-wheel type, alternating-current generator. Each generator will have mounted on its shaft an exciter having a capacity sufficient for its excitation.

The Eureka Stone & Ore Crusher Company, of Cedar Rapids, Iowa, has just published a new catalog descriptive of their stone crushing machinery and equipment. Information is given on the different uses to which the Mitchell "improved crusher" can be put, which is made in fifteen different sizes with capacities ranging from one to thirty tons an hour. The Mitchell "Eureka crusher" is also described, being a machine of all-steel construction, designed to reduce 3-in. and 4-in. rock or ore to sand in one operation. The Eureka Stone & Ore Crusher Company equips plants complete with the necessary elevators, screens, bins and dump cars. They have just recently located in a large new factory building to enlarge their facilities for manufacturing.

The **Ideal Electric & Manufacturing Co.,** of Mansfield, Ohio, announces that the Machinery and Supply Company, of 61 Freemont Street, San Francisco, Cal., has been appointed its Western agent. The Machinery and Supply Company now carry a full line of "Ideal" apparatus, including alternating and direct-current generators, induction motors, both of the slip-ring and squirrel-cage types, special elevator motors and also a complete line of direct-current motors.

Spring Fitting Furnaces.—The W. S. Rockwell Co., of New York City, has issued an illustrated pamphlet of the Rockwell spring-fitting furnace which is designed to heat the steel uniformly and cleanly and to meet the convenience of the fitters so as to turn out a large number of springs per day with the least hard work. The furnace can be operated either with oil or gas fuel.

The **Golden West Pump Co.,** of San Francisco, Cal., has issued a leaflet giving prices and capacities of centrifugal pumps, also tables stating power required for operating. The pumps are intended principally for irrigation and are listed in sizes from 1 in., 25 gal. per min. to 50 in., 100,000 gal. per min. and for heads from 5 to 75 ft. An interesting method of rating is given for irrigation work, for instance, the 12-in. size delivering 4500 gal. per min. will cover 1 in. deep in one hour 10 acres of land; or 6 in. deep in 100 days, pumping 12 hours a day, 44,444 acres.

Pumping Machinery.—The Deming Co., of Salem, O., has issued a new and handsomely illustrated "Catalog H" describing in 190 pages their line of pumping apparatus, including single and double-acting triplex pumps for various services, deep-well power-working heads, artesian well cylinders, rotary and centrifugal pumps. The standard Deming pumps for general service are triplex and arranged for belt, electric, gas engine or water-wheel drive. Direct-acting steam pumps are not shown as the separate-driven type is regarded as much more efficient. They are also self-contained and require no special foundation. A number of artistic illustrations show the pumps in operation in mine and irrigation work, boiler feeding, paper and pulp mills and in general industrial work, in brine circulating, hydraulic pressure accumulators, fire and water supply service, artesian wells, etc. One of the special designs which has found favor with mine managers is a portable, electrically driven triplex pump which is permanently mounted on a mine truck and readily hauled to the section of the mine which needs pumping out. In addition to the pumps a line of small compressors is shown and valves and fittings are illustrated. The catalog also contains some useful technical information, data on compression, capacities of pumps, tables of power, transmission and friction losses and other hydraulic calculations.

Messrs. **Tate, Jones & Company,** of Pittsburgh, have received from the Mogollen Gold & Copper Company, of Cooney, N. M., an order for oil-burning equipment to be applied to their boilers. Their plant is 90 miles from a railway.

Personal

Mr. **H. B. Abbott** has assumed the position as Western representative of this journal, with headquarters at Pittsburgh.

Mr. **Victor M. Braschi,** of Mexico City, was in New York recently.

Mr. **Max Drott,** metallurgist for the Krupp Co., Germany, was in New York recently en route to the mining districts of Central America.

Mr. **Ernest Gayford,** secretary of the General Engineering Company, of Salt Lake City, recently made a trip through Montana and Idaho.

Mr. **John Gross,** of Draper & Gross, Denver, is on a professional trip to Sonora, Mexico.

Mr. **Edmund B. Kirby,** St. Louis, Mo., recently made a trip to the Skeena River district in British Columbia.

Mr. **Mark R. Lamb** is on an extended professional trip through South America for the Allis-Chalmers Company.

Messrs. **James M. McClave** and **Warren B. Lippincott** have recently entered into partnership and opened offices in the Ideal Building, Denver, where they will engage in practice as metallurgical and chemical engineers, giving special attention to the treatment of complex ores. Both of these gentlemen have been identified with the zinc industry of the West for some years.

Mr. **Kenneth Seaver** has been appointed chief engineer of the Harbison-Walker Refractories Company, Pittsburgh.

Dr. **Edgar F. Smith,** the distinguished professor of chemistry and electrochemistry at the University of Pennsylvania and for a series of years vice-provost of the university, has been elected provost on the retirement of Mr. Harrison.

The many friends of Mr. **Frank J. Tone,** works manager of the Carborundum Company of Niagara Falls, will be delighted to hear that he is now quickly recovering from his recent serious illness. He is now going to Colorado or New Mexico for the winter and expects to be back at work in Niagara Falls about May first.

Mr. **G. E. Webber,** who has been in the Transvaal for 17 years, has resigned his position as manager of the Rand Mines, Ltd., and will return to California.

Mr. **O. F. Westlund,** formerly manager of the Aguascalientes plant of the American Smelting & Refining Company, has resigned that position and is now general manager of the Mines Management Company, of New York, with offices in Mexico City.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (Continued).

688,364, Dec. 10, 1901, Edward R. Taylor, of Penn Yan, N. Y.

Resistance or arc type, especially for the production of carbon bisulphide. The furnace comprises a vertical stack of iron lined with fireclay, the lower portion being enlarged and receiving two or more pairs of horizontal opposed carbon electrodes. The electrodes pass through packing glands, and are insulated with alternate layers of mica and asbestos. In operation, coke or broken carbon is fed down through hoppers onto the electrodes to protect the latter from excessive wear. Arcs may be sprung between the electrodes, or enough broken carbon may be fed in between them to serve as a resistor. The central shaft of the furnace is charged with charcoal. Surrounding the middle portion of the shaft is an annular chamber or jacket which is charged with sulphur. The lower enlarged portion of the furnace is also surrounded by four concentric annular chambers receiving sulphur. In operation the sulphur melts and feeds into the zone of heat, there becoming vaporized and rising through the charcoal and forming carbon bisulphide vapor, which is led from the top of the stack to a condenser. The sulphur being a non-conductor can be used to regulate the electric current by varying its rate of supply. The charge ingredients in the annular chambers intercept the escaping heat and return it to the working zone.

688,393, Dec. 10, 1901, Ramon Chavarria Contardo, of Sèvres, France.

Arc type. The furnace has the general shape of the ordinary Bessemer converter, or of a converter having two pouring mouths at opposite sides. The refractory walls are cooled by inclosed pipes carrying water or air. The trunnions are hollow and receive the opposed electrodes, which pass through water-cooled stuffing boxes. The charge, introduced through the mouth or mouths, is distributed over the whole of the bed and under the arc by tilting the furnace to an angle dependent upon the size and shape of the component grains. If the charge becomes pasty and does not descend freely the furnace is tipped further. Charging and smelting is continued until the molten mass reaches nearly to the electrodes, whereupon the furnace is tipped to pour. The single-mouth converter has a tap hole in its rear end. During fusion the exposed walls and tap hole are raised to a temperature often exceeding that of the molten material, superheating the latter as it runs over them when the furnace is tilted to pour.

688,861, Dec. 17, 1901, Charles Albert Keller, of Paris, France.

Resistance type. The two electrodes or terminals are carried by carriages, movable to and from each other by cables and pulleys. Each carriage comprises a vaulted floor, carrying four square blocks of carbon, connected to the electric mains by thin copper sheets. Small carbon sticks projecting upward from the blocks, and the spaces around them, are filled with refractory material, giving an inclined surface. Between the carriages is a truck having a brick hearth to receive the charge. The carriages and truck are inclosed and covered by a chamber having a stack for escaping gases. In operation both the carriages and hearth are covered with loose carbon, which acts as a resistor. When the carriages are drawn apart the charge breaks at the junction of the carriages and hearth in the zone of greatest heat and where the charge has reached the highest degree of fusion. In the production of chrome iron, to prevent its carburization, the carriages and hearth may be covered by a layer of particles of iron, acting as a resistor.

NEW BOOKS.

Allen, J. R., and **Bursley, J. A.** Heat engines: steam, gas, steam turbines, and their auxiliaries. 288 pages, illustrated. Price, $3. New York: McGraw-Hill Book Company.

Carhart, H. S. Physics for college students. 623 pages. Price, $2.25. Boston: Allyn & Bacon.

Ennis, W. Duane. Applied thermodynamics for engineers. 438 pages, diagrams, 316 illustrations. Price, $4.50. New York: Van Nostrand Company.

Kohlrausch, Friedrich. Gesammelte Abhandlungen. Edited by W. Hallwachs, A. Heydweiller, K. Strecker, P. Wiener. Vol. I. Elasticity, heat, optics, absolute electric measurements and miscellaneous. With the portrait of Kohlrausch as frontispiece. 1108 pages, 1 table and 117 illustrations. Price, bound, 27 marks (in New York, $9, retail). Leipzig: J. A. Barth.

Mineral Industry for 1909. Vol. 18. Edited by W. R. Ingalls. 878 pages. Price, $10. New York: McGraw-Hill Book Company.

Ostrup, J. Christian. Standard specifications for structural steel, timber, concrete and reinforced concrete. 99 pages, illustrated. Price, $1. New York: McGraw-Hill Book Company.

Stoddard, J. T. Introduction to general chemistry. Price, $1.60 net. New York: Macmillan Company.

Wellcome Photographic Exposure Record and Diary, 1911. (This well-known annual appears in time for Christmas, neatly bound in red cloth, and contains again a great deal of useful information on developing and printing, but especially on the problem of correct exposure.) 271 pages. Price, 50 cents. New York and London: Burroughs, Wellcome & Company.

BOOK REVIEWS.

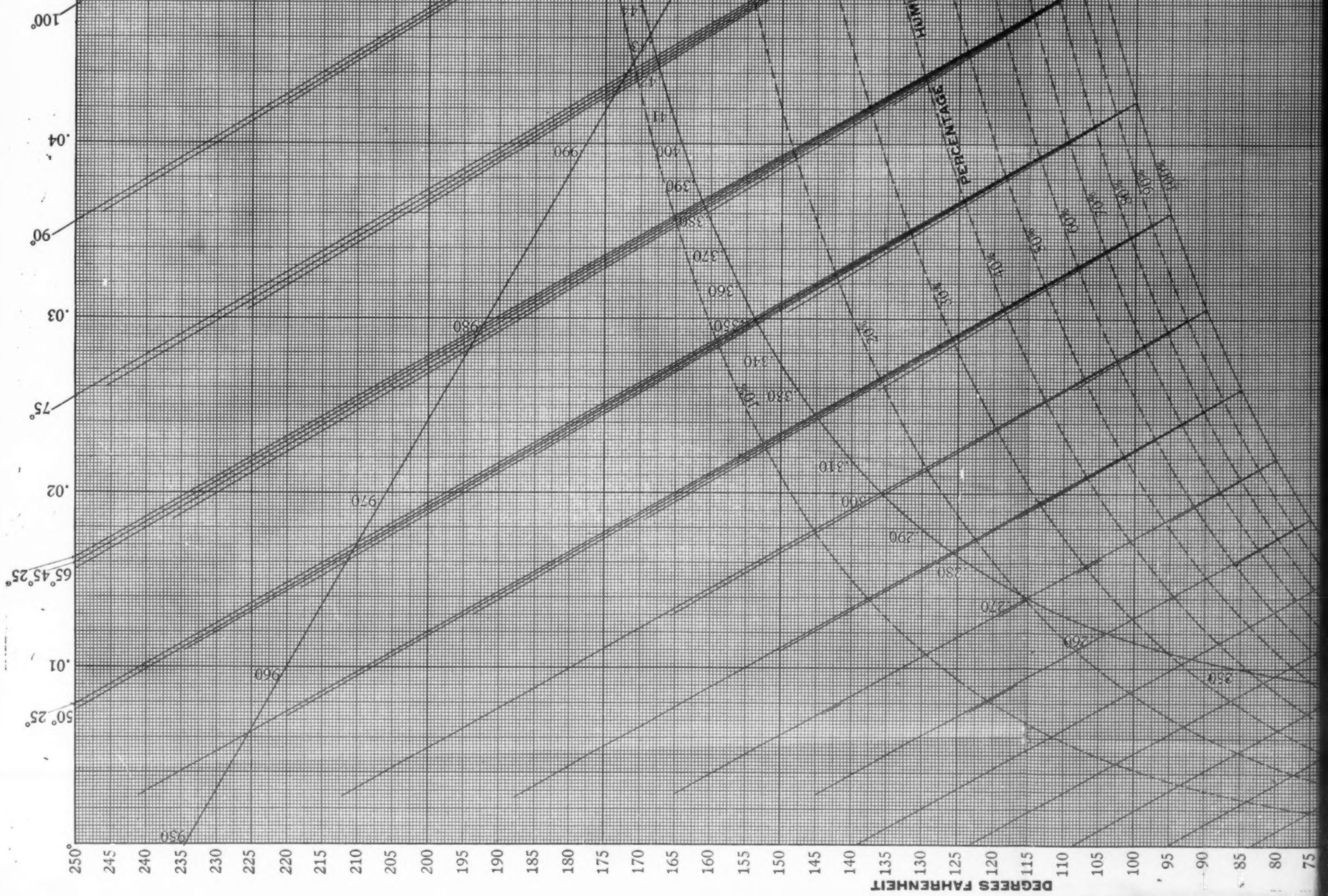
Technologie der Holzverkohlung unter besonderer Berücksichtigung der Herstellung von sämtlichen Halb und Ganzfabrikaten aus den Erstlingsdestillaten. By **M. Klar,** chief chemist of F. H. Meyer, engineers for wood distillation plants. Second edition (1910). 430 pages, 49 illus. Price in New York, $3.70. Berlin: Julius Springer.

Wood distillation has become a very important chemical industry and is still rapidly growing. But exact knowledge of the engineering features and of the commercial conditions under which financial success may be hoped for was for a long time restricted to a comparatively small number of specialists.

Mr. Klar's book, the first edition of which appeared in 1903, was the first reliable modern work based on practical experience and giving exact data. The new edition has been practically rewritten to give to the work more the character of a handbook, and it has been considerably enlarged to cover the progress which has been made in the past six years in the methods of wood disillation and construction of apparatus. In this way a thoroughly up-to-date handbook has been produced based on the practical experience of the author.

There are 20 chapters, dealing with the following subjects: History of wood distillation; raw materials; the chemical changes in wood during distillation; products of distillation; condensing gases: their composition, properties and uses; wood vinegar and its composition; tar and its constituents; charcoal; preparation of wood for distillation; distillation apparatus for large pieces of wood, for wood scrap and sawdust; apparatus for cooling the products of distillation; treatment of non-condensing gases; manufacture of crude wood alcohol and acetate of lime; manufacture of acetic acid and acetone; tar oils; formaldehyde; production of charcoal briquets; analysis of the various substnces; list of patents.

Mr. Klar's book may be sincerely recommended to anyone interested in any way in wood distillation.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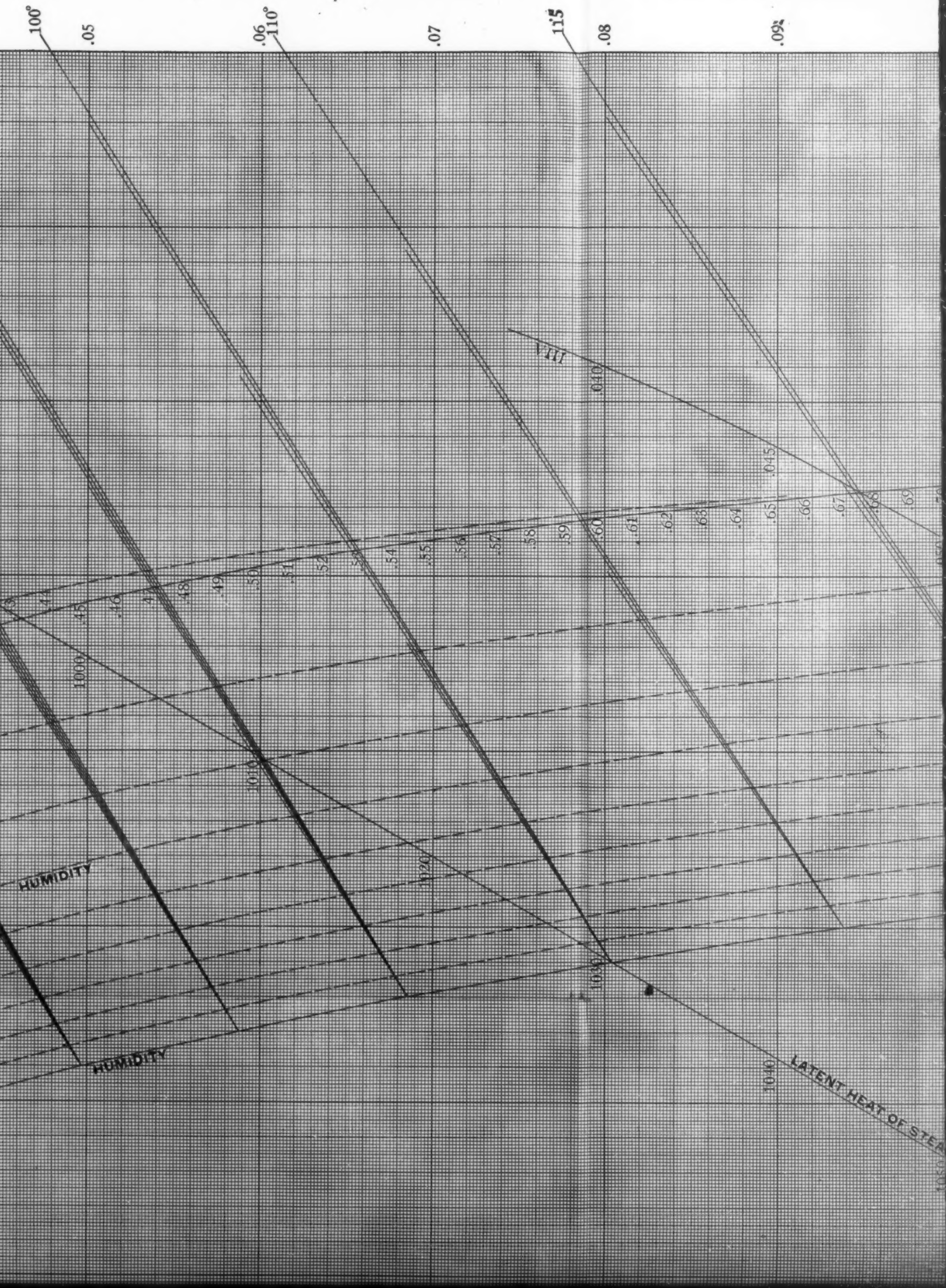
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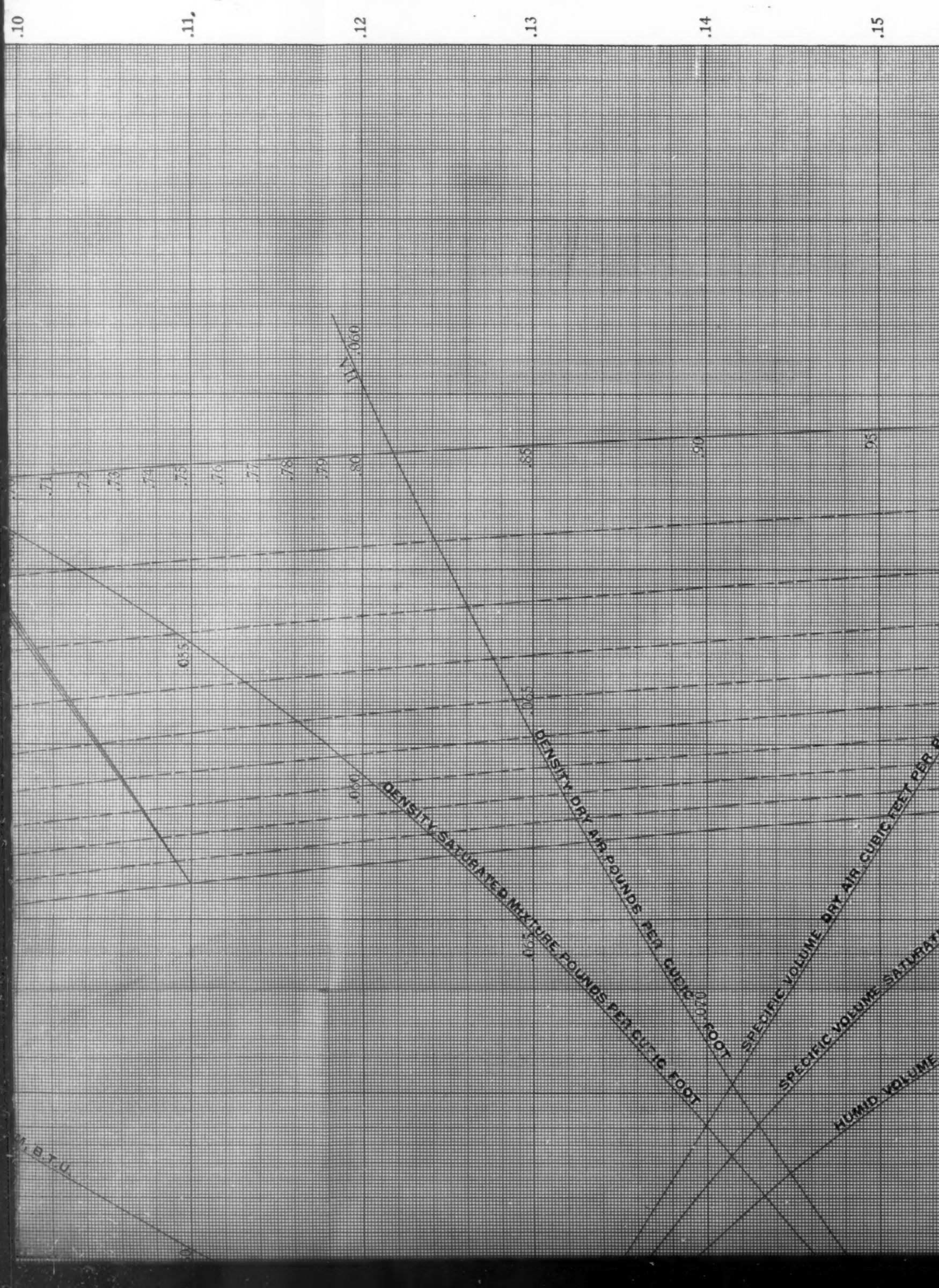
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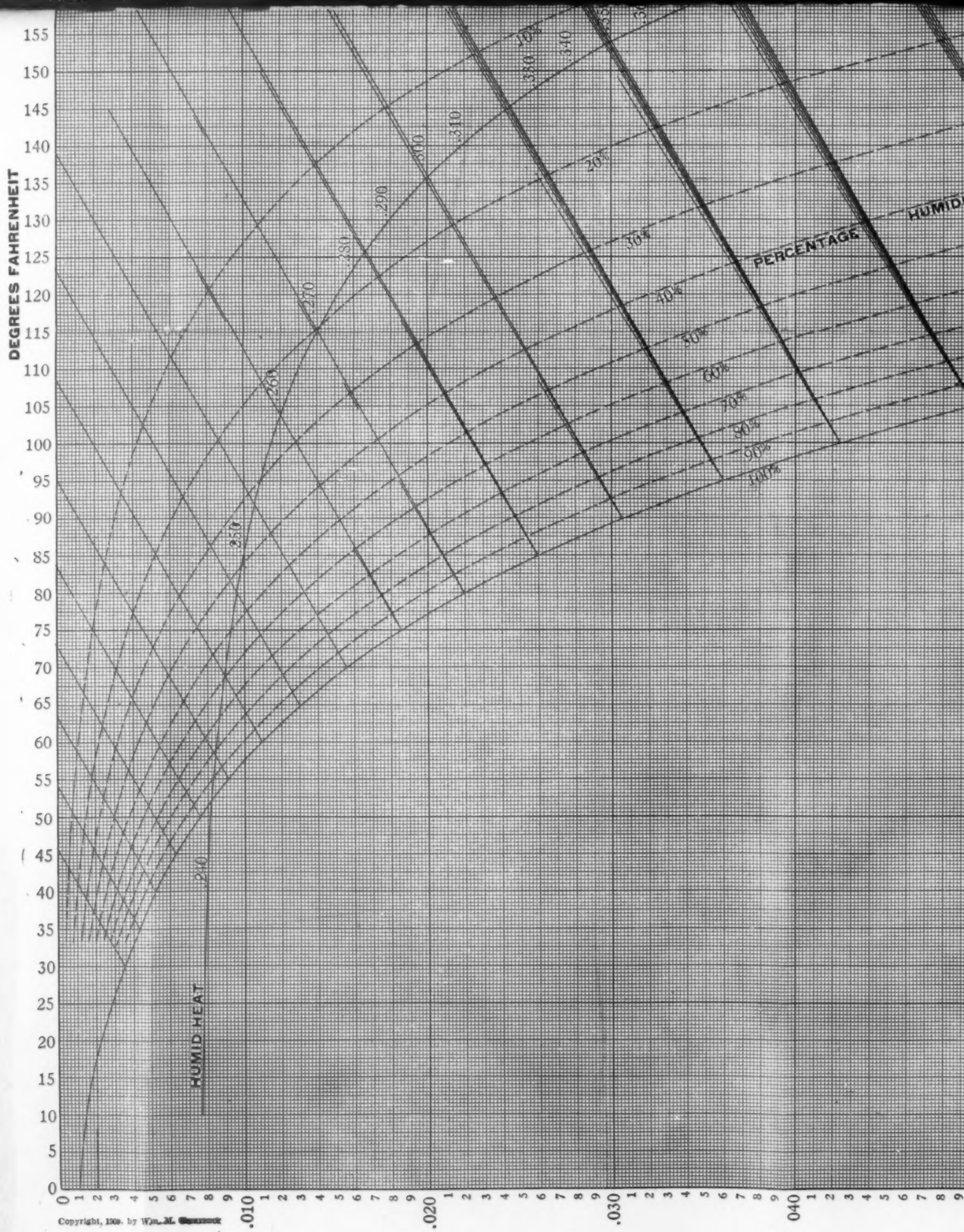

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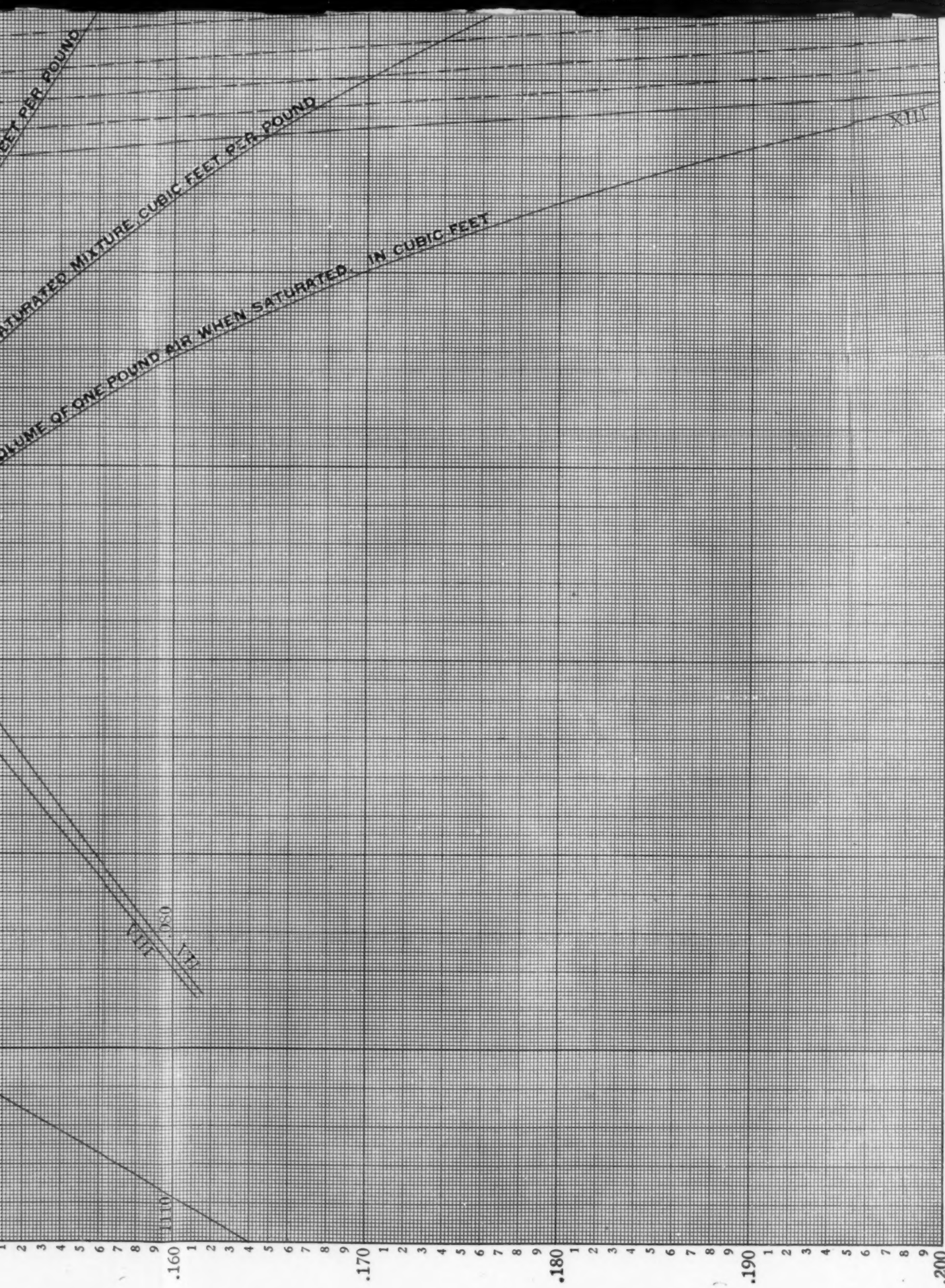

Accompanying "Calculations for Dryer Design" by WILLIAM M. GROSVENOR.